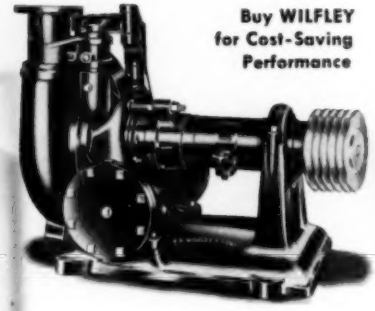
MINING ENGINEERING

MARCH 1951

For mill pulp

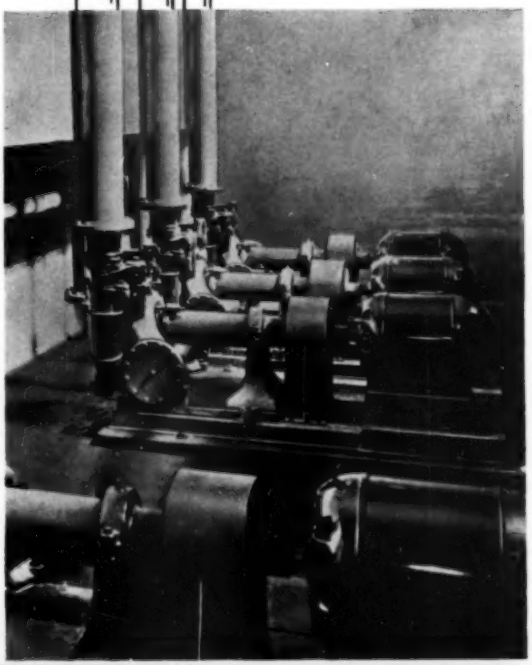


Buy WILFLEY
for Cost-Saving
Performance

WILFLEY
centrifugal PUMPS

Companion to famous
WILFLEY Acid Pumps

A large Western copper mill depends on this bank of 8" WILFLEY Rubber Lined Sand Pumps for handling mill pulp on an economical basis. ⟶



Mill pulp, sands, slimes, or slurries of any kind—all are handled efficiently and economically by famous Wilfley Sand Pumps. And now the new WILFLEY Model K Centrifugal Sand Pump offers many important mechanical improvements which give you even greater dollar savings—in power and production. This rugged, heavy-duty new pump delivers continuous, trouble-free performance, stepped-up production and dependable service. Individual engineering on every application—an economical size for every pumping requirement. Write or wire for details.

●

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Denver, Colorado, U.S.A.

New York Office: 1775 Broadway, New York City

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MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

VOL. 3 NO. 3 MARCH, 1951

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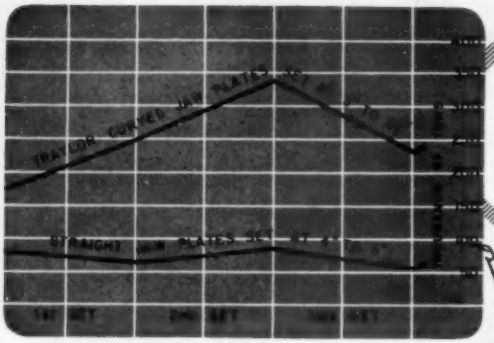
To conform with Postal Regulations, a change had to be made in the method of numbering the volumes of MINING ENGINEERING, beginning with the March, 1951 issue. This issue will be numbered Vol. 3, No. 3; and numbering will be consecutive thereafter. The AIME Transactions, Mining Branch, in a bound volume, will be numbered separately.

Traylor Curved Jaw Crusher Plates outlast ordinary plates almost 3 to 1



This chart is based on actual records kept on a 36" x 48" jaw crusher. Operating at a setting of 4" to 5", the average life of straight jaw plates was 96,500 tons. When fitted with Traylor smooth-faced, *curved* jaw plates, this same crusher averaged 276,724 tons per set . . . *at a setting of 3" to 3½"*. Plate life was increased almost 3 to 1!

No choking . . . no packing . . . stops power waste. Crushing force is used with greater efficiency to produce a more uniform product with fewer waste fines.

Traylor Curved Jaw Crusher Plates . . . standard equipment on all Traylor Jaw Crushers . . . can be installed in many other crushers to transform them into modern, up-to-date primary breakers. Traylor's Curved Crushing Surfaces cut costs of primary stone reduction in *many* ways . . . outlast ordinary plates as much as 3 *to* 1. If you want to increase the efficiency of your plant, investigate Traylor Curved Jaw Plates today.

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Rotary Kilns, Coolers and Dryers · Grinding Mills
Jaw, Reduction and Gyratory Crushers · Crushing Rolls

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to get full details.

TRAYLOR ENGINEERING & MANUFACTURING CO.
325 MILL ST., ALLENTOWN, PA.

I want to see how easily Traylor Curved Jaw Plates can be installed in my present crusher.

*Name*_____

*Company*_____

*Address*_____

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SALES OFFICES: New York, N.Y.; Chicago, Ill., Los Angeles, Calif.
Canadian Mfrs: Canadian Vickers, Ltd. Montreal, P.Q.

A "TRAYLOR" LEADS TO GREATER PROFITS

Here's something more . . .

Case-histories No. 1 and No. 2 (previously published in this series) gave service tests involving the crushing of millions of tons of ore. Here are further cases taken from actual ore-grinding practice—you can rely on these data.

Write for 32 pp reprint of an informative paper published by the Amer. Inst. of Mining and Metallurgical Engrs., giving results of wear tests on grinding ball materials—FREE!

Case No. 3

Tests in Colorado, 1939: Molybdenite ore—quartz and feldspar, principal abrasives.

Service: "98" Marcy grate discharge type ball mills grinding ⅜" siliceous ore to minus 100 mesh. Mill speed 20 rpm: balls charged were 3" dia.

Comparison: Lifter type clamp bars in low discharge grates, over a six-months period. Four bars of austenitic manganese steel together with four bars of chromium-molybdenum steel, normalised and tempered to 375 brinell. The bars were not worn flush with the grates when they were removed for weighing.

Results: Four manganese steel clamp bars lost 166lb. weight

Four Cr-Mo steel clamp bars lost ... 144lb. weight

In this test the manganese steel bars wore 15% faster than the chrome-molybdenum steel.

Case No. 4

Tests at a mine grinding low-grade copper ore, 1945-1949.

Service: Marcy 10′ × 10½′ grate discharge type ball mills grinding minus ¾" feed to 84% minus 100 mesh. Mill speed 18 rpm: balls charged were 2½" and 2" dia.

Comparison: Several sets of austenitic manganese steel liners, concurrently with one set of high carbon chromium-molybdenum steel liners normalised and tempered to 350 brinell.

Liner Type: Shell (barrel) liners, ship lap type; weight 47,000 lb. approx. per set.

Results:

	Tonnage ground	*Life-Days*
Manganese steel, average...	593,344	337
Cr-Mo Steel	667,308	379

The chrome-molybdenum steel liners wore 12% longer than the manganese steel liners.

Climax Molybdenum Company

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Please send your FREE Reprint on "Wear Tests"

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Position .....

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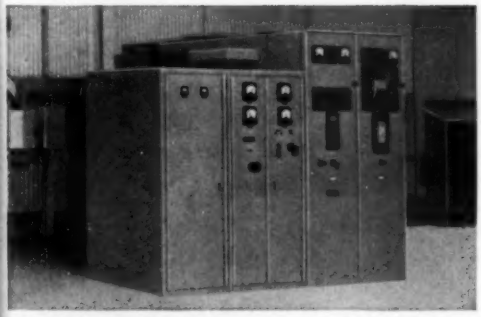
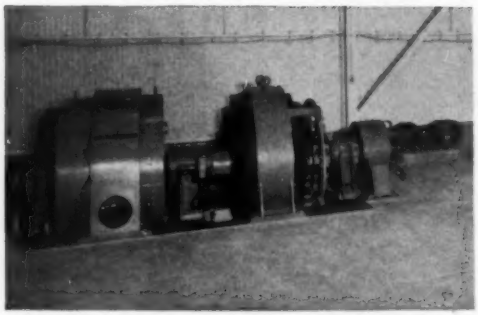
MOLY

AUTOMATIC HOISTING for continuous tonnage!



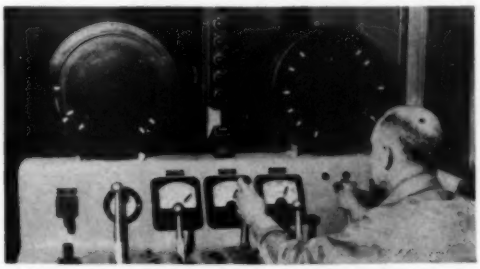
Completely metal-enclosed for personnel safety, this control cubicle houses the synchronous motor switchgear panel, plus Ward Leonard type d-c control equipment. Less space requirement and simplified control result from use of the G-E amplidyne, which eliminates need for many contactors and relays, while permitting the retention of safe limits on the hoist's acceleration and deceleration.



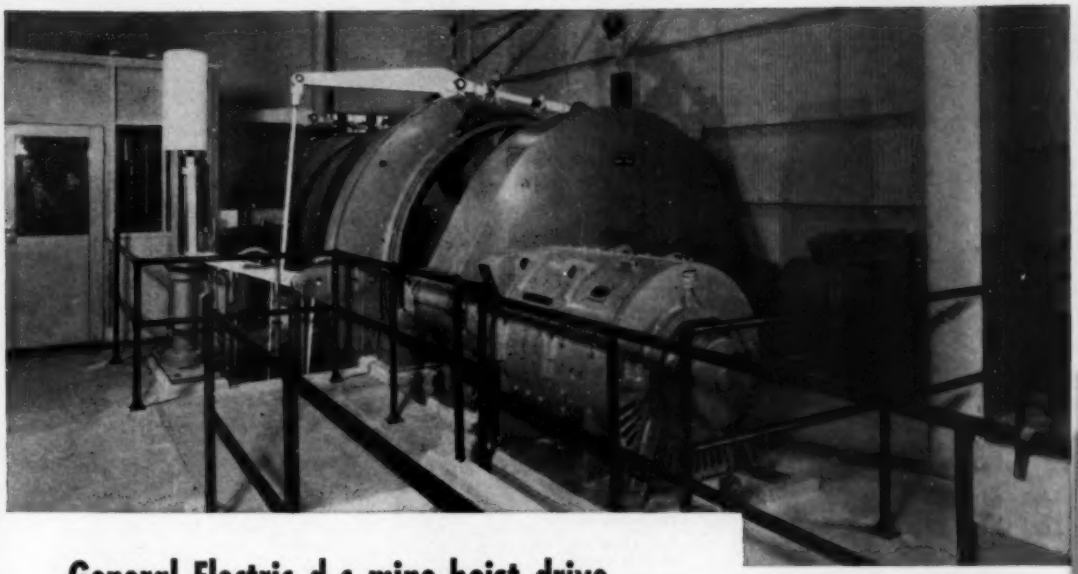
Converting a-c to d-c, this G-E motor-generator set comprises a 750-kw generator, an 1100-hp synchronous motor, and a 20-kw exciter. The generator supplies adjustable-voltage power for the operation of the two hoist motors. At right, the G-E amplidyne, used as an exciter for this generator, provides superior operation and more positive limits of acceleration and deceleration, for greater safety, increased production.

MINE-HOIST DRIVES

—to cut mining costs!



When necessary for inspection, testing, or man-trips, control of the hoist can quickly be switched from automatic to the manually operated master controller as above. Adequate starting, stopping, and maneuvering of the skips with ease, precision, and safety is provided by the optional manual operation. With the normal automatic operation, the duties of the operator may be chiefly of another nature.



This automatic mine hoist is powered by a complete G-E drive that includes two 500-hp d-c motors. When the skip at the shaft bottom is fully loaded, it starts and accelerates to full speed. Near the dumping pocket, it automatically slows down, eases through the dumping horns, and makes a dead stop at the final limit of travel, thus completing the entire cycle automatically.

General Electric d-c mine-hoist drive with amplidyne control permits continuous output at 8 tons per minute, simplifies operation, increases safety, prolongs equipment life!

Here is another General Electric amplidyne-controlled d-c mine hoist drive helping to boost output and cut costs—in this case in a New Mexico potash mine.

This high-speed high-tonnage hoist raises in balance an 8-ton payload every minute up an 1150-foot shaft at speeds up to 1500 fpm. The hoist runs fully automatically throughout the entire shift, and may also be run by push-button control operated by the skip loader at the loading level.

Because it is completely automatic, this continuous, highly efficient operation permits hoisting maximum tonnage per man-hour. Hoist speed is closely maintained in spite of load variations, repetitive operation helps cut costs per ton, and personnel may be freed for other duties. Electric braking provides greater safety and prolongs the life of the hoist mechanism.

Applicable for any type of mining, this installation is further evidence of G-E experience in engineering automatic mine-hoist drives, experience that dates back to 1915. Let a G-E mining specialist put this experience to work on *your* mine-hoist problems. Call him at your nearest G-E office. Meanwhile, send for Bulletin GET-1430. *Apparatus Dept., General Electric Company, Schenectady 5, N.Y.*

GENERAL ELECTRIC

660-22

Personnel Service

THE following employment items are made available to AIME on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York Office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Recent Graduate, 30, with B.S. degrees in mining engineering and geology, desires position as junior engineer or geologist. Five years' practical mining experience before college. College record considerably above average. Go anywhere. Available immediately. M-607.

Mill Superintendent, 43, married, two children, good engineering background and wide experience in milling all types of ores, metallic and nonmetallic. Capable of assuming administrative position relative to mining and milling. Prefer southwest United States but will consider other locations. Available after Apr. 1. M-608.

Graduate Mining Engineer, 40, B.S. in civil engineering and an engineer of mines degree; married, two children. Eighteen years' experience anthracite coal and phosphate rock open pit mining. Five years' experience general superintendent and qualified for executive responsibilities and duties. Available immediately. M-609.

Geologist, Ph.D., 32, married; two field seasons as chief of geological parties in Quebec; one field season as chief geologist for company on Labrador Range; one year professor of geology; two years geologist examining occurrences of raw materials need in the production of steel. Location, Canada or United States. M-604.

POSITIONS OPEN

Superintendent with placer experience, especially experienced in operating a dragline, dryland placer unit. Preferably experienced with tin and its recovery by Pan American Placer Jigs. Company is preparing to put into operation a movable land placer washing and cleaning plant and mill separation circuit. The metals to be recovered are tin, gold and tungsten in the form of scheelite and the concentrates separated on the property. Location, South Dakota. Apply by letter giving full particulars and salary requirements. Y4930.

Mining Engineer with considerable experience in mining of coal and hard rock. Two-year contract. Salary, $13,000 a year plus expenses. Location, Formosa. Y4929.

Junior Mine Engineer, recent college graduate, competent underground surveyor and draftsman. Standard three-year contract, single, free transportation furnished by air, four weeks' vacation yearly plus free living quarters. Salary, $3300 a year plus yearly bonus. Location, Bolivia. Y4872.

Geologists and Mining Engineers. (a) Senior Geologist with at least five years' field experience covering reconnaissance and detailed general studies of mineral deposits. (b) Junior Geologist for field work covering aerial, surface and sub-surface mapping in metal mining field. (c) Mining Engineer with at least five years' exploration development and survey experience, to analyze projects, prepare reports and supervise developments. (d) Junior Mining Engineer with at least two years' surface and underground experience, for general mine prospecting and development work. Salaries: (a) $4600 to $6500 a year; (b) $3100 to $3800 a year; (c) $6000 to $7000 a year; (d) $4000 to $5000 a year. Location, western United States. Y4856.

Engineers. (a) Chief Geologist, with thorough academic grounding, a number of years' responsible experience. Salary, $6000 a year. (b) Assistant Business Superintendent, preferably a graduate of recognized university with an accounting major. Must have working knowledge of Spanish. Should be a capable and experienced accountant. Salary, $6000 a year. Transportation and traveling expenses paid for employee, wife and one child, round trip. Hospitalization, medical service, and housing free. Three-year contract. Location, Bolivia. Y4825.

Engineers. (a) Designer with considerable experience on the processing and layout of mining and smelter equipment. (b) Draftsman with mine and smelter experience, to lay out plant and machinery. Salaries open. Location, New York City. Y4808.

Junior Geologist, recent graduate, or with one or two years' experience for exploration work in Southeast. Salary open. Headquarters, Delaware. Y4765.

Mill Shift Bosses for a gold mine in Colombia, S. A. Must work with native labor, and have a knowledge of Spanish. Salary, $2700 a year for three-year contract. Y4558.

Metallurgical or Chemical Engineer, young, mining graduate, with considerable experience in ore flotation or dressing for the separation of various minerals. Location, Indiana. Y4521C.

Chemist-Assayer to do fire assaying and inorganic chemical analyses on a very wide variety of ore samples in connection with an ore dressing research laboratory. Salary, $4781 a year plus transportation. Location, Alaska. Y4515.

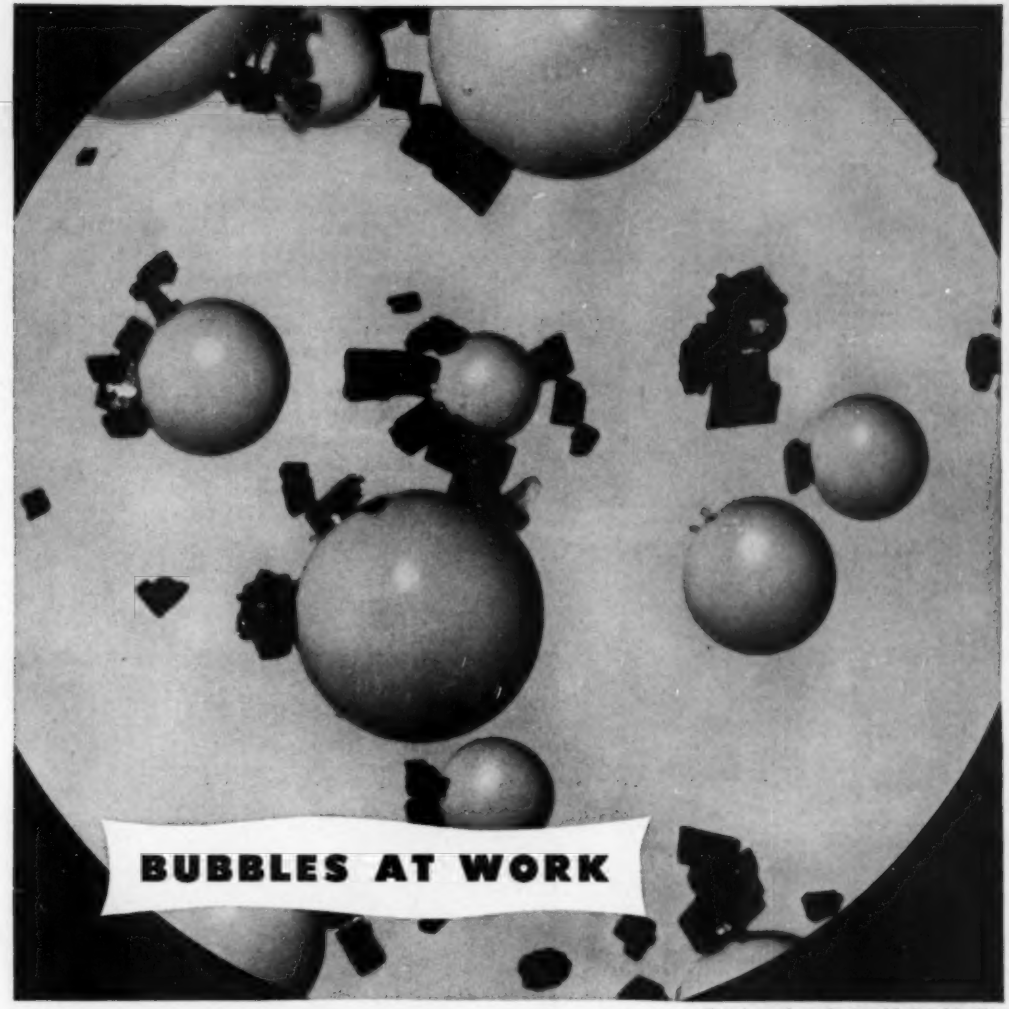
JUNIOR MINE ENGINEER. Recent college graduate, competent underground surveyor and draftsman, standard three-year contract, single, free transportation furnished to Bolivia by air, four weeks' vacation yearly, plus free living quarters, salary $3300 yearly, plus yearly bonus one month's salary.

Box C-6, MINING ENGINEERING

Senior Metallurgists

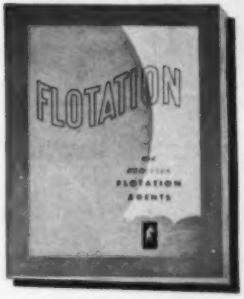
Positions available for appointment as senior metallurgists for service in Northern Rhodesia to supervise special research projects on base metal flotation and leaching problems. Applicants must be technically qualified and have minimum five years' responsible industrial experience. Salary and conditions of engagement to be negotiated according to qualifications. Box No. 576/4, Foster Turner & Everetts Ltd., 11, Old Jewry, London, E.C.2.

Assistant Manager. Position available for assistant manager to supervise metallurgical operations of large copper enterprise including concentration, smelting, and refining. Applicant must be technically trained with good organizing ability, familiar with new developments, and with practical experience in all phases copper metallurgy. Preferred age 40 to 50. Location Northern Rhodesia. Salary to be negotiated according to qualifications and experience. Box No. 573/5, Foster Turner & Everetts Ltd., 11, Old Jewry, London, E.C.2., England.



Photomicrograph, greatly enlarged, by H. Rush Spedden

To help step-up flotation recoveries at low cost, Hercules has developed a number of frothers and collectors. Yarmor® F Pine Oil has been a standard of quality among pine oil frothers since the early 1920's. RADA (Hercules' Rosin Amine D Acetate) is a siliceous mineral collector that has good solubility and is easy to handle. It is used in the commercial flotation of several nonmetallic ores. These and other Hercules products are described in a 16-page technical booklet, "Flotation and Hercules Flotation Agents." This booklet also contains many formulas useful in the operation and control of modern ore-dressing mills. *Send for your copy.*



HERCULES POWDER COMPANY 955 King St., Wilmington, Delaware

HERCULES Flotation Agents

NM51-1

you can stop dry drilling with a GARDNER-DENVER RB STOPER

An exclusive Gardner-Denver feature can help you eliminate dry drilling. It's the new *air-operated* water control valve — available only on Gardner-Denver RB Stopers.

This unique valve assures a flow of water to the face of work both before and during the drilling operation. *One* throttle valve, operated by a conventional throttle valve handle, controls all functions of drilling and water control. The desirable "water on—air on—air off—water off" cycle required in some localities is provided automatically.

and here's another RB advantage you'll like

The throttle valve is fitted with a tubular screen which is automatically cleaned by live air each time the handle is brought past the "off" position. Any accumulated dirt is blown out of the drill, and time out for cleaning a clogged screen is practically eliminated.

Gardner-Denver RB104 —the heavy-duty self-rotating stoper with exceptionally high drilling speeds.

Gardner-Denver RB94 —the fast drilling, self-rotating stoper that weighs only 100 pounds.

Gardner-Denver Stopers are famous for their safe handling, fast drilling, smooth feeding and their constant sludge protection.

Bulletin SD2 gives complete information on Gardner-Denver Stopers —write today for your copy.

GARDNER-DENVER *Since 1859*

Gardner-Denver Company, Quincy, Illinois
In Canada:
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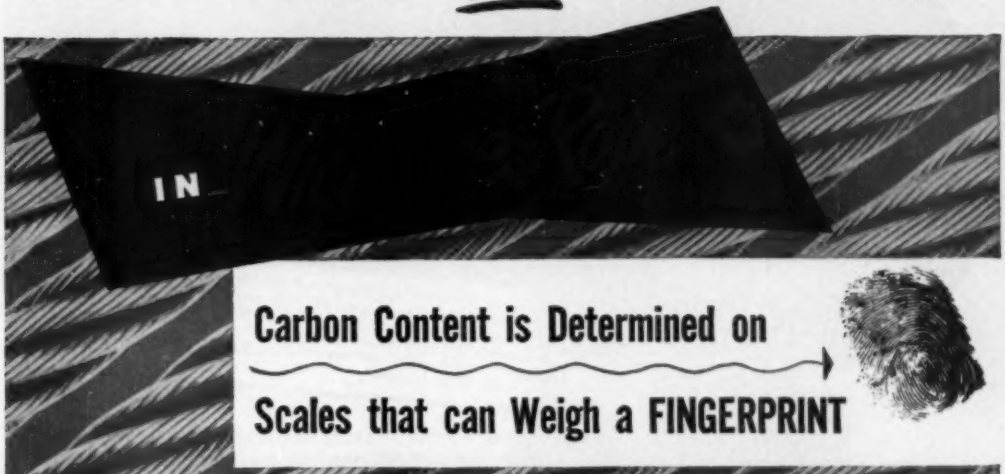
ALL ROPES *look* ALIKE... but



Carbon Content is Determined on Scales that can Weigh a FINGERPRINT

Scales so sensitive they can weigh your finger print—that's a typical example of the exacting control we maintain over steel that goes into Wickwire Rope.

Because of such precision analysis and testing, we're able to hold and maintain carbon content of steel within five points (.05%) of the appropriate value for each of our commercial grades of wire rope. That's mighty important—because correct carbon content of steel determines its strength, toughness, ductility, hardness and uniformity of grain size to McQuaid-Ehn* standards.

Basic control like this is possible only with a fully integrated company like Wickwire where manufacture of wire rope starts with the actual making and refining of the steel.

Thus, Wickwire goes "beyond specifications" to supply you with wire rope that has no equal for longer life, utmost safety and enduring reliability.

Look for the YELLOW TRIANGLE on the reel

*Write for detailed information on the McQuaid-Ehn test to Wire Rope Sales Office, Palmer, Mass.

WICKWIRE ROPE

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A PRODUCT OF THE WICKWIRE SPENCER STEEL DIVISION OF THE COLORADO FUEL AND IRON CORPORATION

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PACIFIC COAST SUBSIDIARY—The California Wire Cloth Corporation, Oakland 6, California

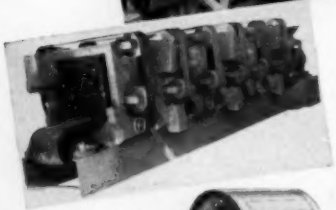
DINGS magnetic equipment for every mining requirement!

Dings Rectangular Electromagnet provides maximum iron removal above belts or chutes. Triple pole design for great magnetic strength and depth of penetration. Maximum magnetic strength across entire magnet face. Easily installed in any position. Low operating cost . . . trouble free . . . complete range of sizes.

(Right) Induced Roll Separator purifying sand.

Battery of Crockett Separators concentrating magnetite. (Left)

Magnetic Detector signals presence of tramp iron. (Right)



NEW, powerful Cross Belt Separator — with higher selectivity—for concentration and purification (Above)

NEW, powerful, non-electric Alnico Perma-Pulley for purifying, cobbing, protecting crushers, pulverizers, etc. Magnetic permanence guaranteed for life of installation. (Above)

Wherever crushers must be protected against tramp iron, wherever magnetic separation can contribute to the beneficiation of ores, you will find Dings magnetic separators specially designed to meet the requirements of the job. Dings magnetic separators have played an important role in making the extraction of many minerals possible and economically feasible. Shown here are a few of the machines which Dings has pioneered exclusively for the mining industry for the following purposes:

CONCENTRATION—everything from specially designed pulleys for cobbing nickel ores to powerful induced roll and cross belt separators for concentrating manganese, tungsten, chrome, titanium and many other minerals.

PURIFICATION—removing small amounts of feebly magnetic impurities with super high intensity separators. Minerals treated include feldspar, silica sand, salt, bauxite, barytes, nepheline-synenite, gypsum.

CRUSHER PROTECTION—magnetic pulleys, pulley type separators, drums, suspension magnets, spout magnets and magnetic detectors are all widely applied for arresting tramp iron ahead of crushing equipment.

WET TYPE SEPARATORS—Dings has pioneered in the development of wet type separators for concentrating magnetite and for use in new processes for concentrating low grade ores. *Ask for complete details of Dings Separators for Heavy Media Process Plants.*

Take advantage of Dings equipment and experience. For practical recommendations, submit your requirements with 25 lb. samples of material to the Dings Laboratory for testing and analysis. Literature on request.

DINGS MAGNETIC SEPARATOR CO.
4718 W. ELECTRIC AVENUE, MILWAUKEE 14, WISCONSIN

"Magnetic Separation Headquarters Since 1899"

Dings
"HIGH INTENSITY"

the fast,

low-cost way

to get

worn teeth

back in service!

1. Worn tooth . . . 2. Plus AMSCO Tooth Repointer . . . 3. Equals like-new tooth—ready for service!

AMSCO® tooth repointers

AMSCO Repointer Bars

3 teeth partially rebuilt with AMSCO Repointer Bars to show sectional view of welding method. These Repointer, or Wedge Bars, are also made in many shapes and sizes. Apply AMSCO Economy Hardface to the rebuilt tooth *—for as much as 3 times longer service!*

● Here's a typical example of a badly worn dipper tooth completely restored to service . . . simply by welding on the AMSCO Cast-To-Size Tooth Repointer. The result is like-new service life and efficiency, plus a long-wearing tooth of Manganese Steel—the steel that actually work-hardens in service. Cost? *Far less than replacement!*

Wherever equipment has teeth that wear, more and more owners are using money-saving AMSCO Tooth Repointers . . . available in a wide variety of styles and sizes.

Get all the facts! Write today for Bulletin W-10-A —showing complete instructions for use.

put LIFE into service with

AMSCO®

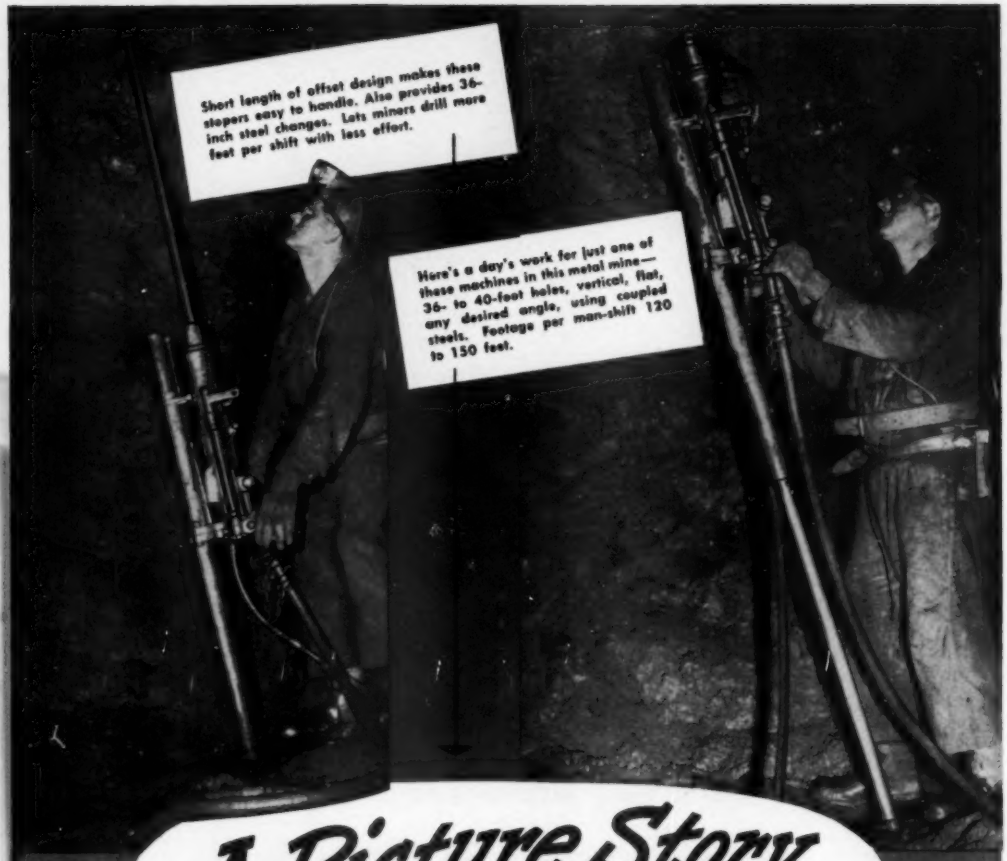
WELDING PRODUCTS

AMERICAN Brake Shoe COMPANY

AMERICAN MANGANESE STEEL DIVISION

417 EAST 14th STREET • CHICAGO HEIGHTS, ILL.

Other Plants: New Castle, Del., Denver, Oakland, Cal., Los Angeles, St. Louis. In Canada: Joliette Steel Division, Joliette, Que.


Short length of offset design makes these stopers easy to handle. Also provides 36-inch steel changes. Lets miners drill more feet per shift with less effort.
Here's a day's work for just one of these machines in this metal mine—36- to 40-foot holes, vertical, flat, any desired angle, using coupled steels. Footage per man-shift 120 to 150 feet.

A Picture Story of Lower Stoping Costs

. . . shows how Le Roi-CLEVELAND offset stopers drill extra footage, save you money

Take a good look at these pictures. Note the length of the drill steel. Fitted with carbide-tipped bits, it gives you 36" of drilling—3 feet with every change.

That's one of the reasons why Le Roi-CLEVELAND offset stopers are paying off in lower drilling costs.

But it's not the whole story. The machines shown here are equipped with downstroke rotation. This gives plenty of power and lets the miners drill holes from 36 to 40 feet in depth — using coupled steel. Total footage drilled per man-shift runs from 120 to 150 feet.

Performance like this does a lot to keep costs down. It also accounts for the growing popularity of Le Roi-CLEVELAND offset stopers wherever stoping is a problem.

Other features of these machines are: Removable water tube — no need to dismantle the machine. Graduated feeding pressure. Constant blowing at chuck keeps machine clean. Write us today for complete details.

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Plants: Milwaukee, Cleveland and Greenwich, Ohio

Abstracts of Technical Papers Appearing in Journal of Metals

During recent months, several articles have been published in JOURNAL OF METALS, the AIME Metals Branch publication, that bear on the activities of mining site operations. These articles are listed below for the convenience of the reader.

Copies of issues in which individual articles appeared can be obtained from AIME at the single copy price of the magazine, or photostats of the articles can be obtained at the regular prices for such work.

October: **The Development of the Lead Blast Furnace at Port Pirie, South Australia.** L. A. White. This paper describes the developments of the lead blast furnace at The Broken Hill Associated Smelters Pty., Ltd., from the original standard furnace to the present wide shaft type, with two superimposed rows of tuyeres. The metallurgical operations of the blast furnace section of the Port Pirie Works are also described.

February: **Reverse Leaching of Zinc Calcine.** L. P. Davidson, R. K. Carpenter, and H. J. Tschirner. The electrolytic zinc plant of the American Zinc Co. of Illinois, at Monsanto, was expanded in 1948 to a capacity of 100 tons of slab zinc daily. This capacity was not attained because of inability of the leaching plant to deliver an adequate amount of solution for electrolysis. Changing the leaching method so that the acid was added to the roasted zinc material reversed the usual procedure and made it possible to attain the desired capacity. The conditions which prevented satisfactory work before this change and the difficulties which arose in reversing the usual leaching procedure are described.

February: **Preparation of Titanium Tetrachloride from Rutile.** C. H. Gorski. A method for preparing titanium tetrachloride is described which consists of reducing rutile with coke and chlorinating the reduced product at 200° to 500°C. The crude distillate is purified by treatment with copper powder and distillation. Recoveries, exceeding 90 pct of the titanium in the rutile, were obtained.

February: **Sintering Practice at Josephtown Smelter.** H. K. Najarian, K. F. Peterson, R. E. Lund. Sintering at the Josephtown smelter differs in many important respects from the sintering practice in smelters operating horizontal retort zinc furnaces. Requirements of the electrothermic smelting furnaces define the physical characteristics of the sinter, while the chemical composition of the sinter is controlled according to the grade of metal and oxide to be made as final products.

at the working face . . .

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Above ground it's free, but down in the bowels of the earth, where men mine the pay-lode or build tunnels for the benefit of humanity, *AIR* and its *DELIVERY COST* are vital factors.

SPIRATUBE-M and AYRTUBE cut Air *Delivery Costs*. Made specifically for mine and tunnel ventilation, these two new *AIRDUCTS* working individually or in combination, deliver a maximum cubic measure of air right at the working face and at the lowest possible cost—because—ease of handling reduces installation costs, and construction features minimize leaks and seepage. You get the full volume created by your fans and blower units right where you need it—at the working face—reducing power costs.

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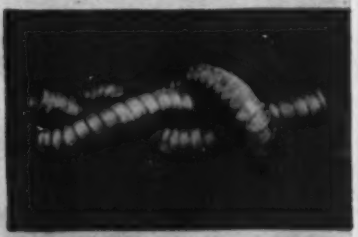

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Remember Eimco's for loading — the list at right represents a few of the successful tunnels loaded out with Eimco RockerShovels.

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- Wingebocking
- Alpine-Draper
- Olmstead
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- Granby Dam
- Rainbow
- Leadville
- St. Marys
- Carter Lake
- Pole Hill
- Rattlesnake
- Olympus
- Chuquicamata
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- Orchard Point
- Tunnel del Tecolute
- Murchie
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IT'S no passing fancy that has caused experienced welders to depend on ANACONDA Welding Rods. These rods, especially Tobin Bronze*, have been their long standing choice where the dependability of the weld metal counts most.

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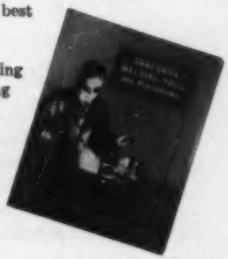

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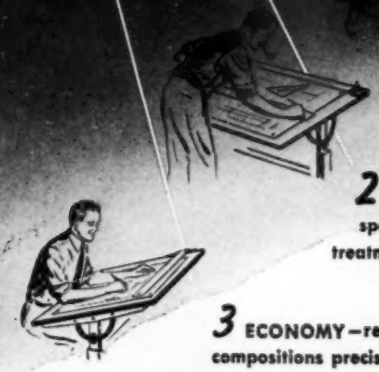

1 PERFORMANCE—Strength and toughness, resistance to wear, fatigue or shock to meet a wide range of requirements, as dictated by design.

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Experience shows that triple-alloy steels containing Nickel are solving some mighty big problems in many industrial fields. They have established outstanding service records in some of the most exacting applications. The many standard compositions available make it possible to select *accurately*, and with economy, triple-alloy steels to fulfill the requirements of a great variety of applications.

We invite inquiries regarding the selection and uses of triple-alloy steels, containing Nickel.

THE INTERNATIONAL NICKEL COMPANY, INC. 67 Wall Street New York 5, N. Y.

* A demonstration pipeline for transporting coal will be constructed by Pittsburgh Consolidation Coal Co. at the Georgetown strip-mining operation near Cadiz, Ohio. The 12-in., 17,000-ft line, special crushing and preparation equipment, and pumps especially designed for the operation will cost about $550,000. A slurry of fine coal and water will be fed into the line and moved through it under pressure.

* Sulphur resources in the U.S. are being rapidly exhausted with no new major deposits being found. This common mineral may become one of the most strategic items during the next five years. Recovery of sulphur from pyrites is being considered in case of an emergency. Already the British have adopted stringent measures to conserve sulphur supplies.

* National Steel Corp. has acquired a 15 pct interest in Reserve Mining Co. The remaining interest is held by Armco and Republic Steel. Reserve was organized to develop processes for the extraction of iron from taconite.

* Deliveries of nickel in all forms in 1950 by the International Nickel Co. of Canada were 256,000,000 lb constituting a record for any peacetime year. Deliveries in 1949 were 209,292,257 lb. Production is currently at maximum and a new record is expected to be set in 1951. The company expects to complete in 1953 its program of full conversion from surface to underground mining.

* Expansion plans of the various steel companies will raise the total annual steel capacity of the United States to a record high of 117,500,000 tons at the end of 1952. More than 18 million tons will have been added in three years from the start of 1950.

* A new process using liquid nitrogen makes it possible to grind the toughest materials to powder. Developed by Linde Air Products Co., the use of the process is limited to high cost materials which are cooled to a point of maximum fragility by a liquid nitrogen spray.

* A new use for mica flakes has been developed by the Westinghouse transformer division which uses it in a three-coat corrosion resistant paint. The mica flakes are used in the middle coating of this triple protective covering, and are embedded in a syrup-like plastic.

* Molybdenum coated articles could well be used as an economical substitute for those made entirely of that metal according to W. J. Childs of Lafayette College, J. E. Cline, W. M. Kisner, and John Wulff of MIT. It may also be useful for coating other metals and alloys.

* Cleveland-Cliffs Iron Co.'s Mather mine "A" shaft produced 1,251,963 tons of ore last year - said to be the largest production ever obtained through a single shaft in any iron mine in the entire world.

Allis-Chalmers Service Does Not Stop When Your Equipment is Installed

YOU SEE, Allis-Chalmers maintains a staff of special repair parts service experts, whose job is to see to it that emergency repair orders are given emergency treatment.

On the alert day and night (they can be reached at home), these men get on the job at once when an emergency call comes through. Working closely with shop management, they locate parts or materials, make sure that shop machinery and machine operators get on the job right away. Most important — *they stay on the job* until the needed part is on its way and actually delivered to you!

Fast, dependable repair parts service is an important reason why many men in industry keep on specifying Allis-Chalmers equipment year after year. As one A-C customer put it, "We know that when Allis-Chalmers repair parts men go to work for us we'll get our parts very soon." A-3284

New Crusher Shaft Delivered Two Days After Old One Breaks!

FRIDAY P. M.
A gyratory crusher goes down with a broken mainshaft in Watertown, N.Y. plant of General Crushed Stone Company, stopping production.

SATURDAY A. M.
Supt. Bovee, of General Crushed Stone, calls Allis-Chalmers Maintenance and Repair Section, explains predicament. An Allis-Chalmers repair parts expert arranges to follow through at once.

SATURDAY P. M.
New mainshaft and necessary auxiliary parts are rounded up in West Allis-Works (where they are held in readiness for just such an emergency) and leave A-C yards at 6:25 p.m. Saturday. Meanwhile, Allis-Chalmers Traffic department is able to speed shipment by making a route change at Chicago, saving 48 hours. Repair parts man calls Supt. Bovee, tells him shipment will arrive Syracuse, N.Y. 6:30 p.m. Sunday, and supplies car number.

SUNDAY P. M.
Truck is waiting at Syracuse to take shipment direct to Watertown, N.Y.

MONDAY
Crew works Sunday night and Monday completing installation of new mainshaft and parts. Plant is back in operation first thing Tuesday morning. (Monday was a holiday, so hardly any time was lost at all.)

SERVICE LIKE THIS *is not exceptional when emergencies exist. It's typical of the repair parts service you can expect after your Allis-Chalmers machinery has been installed. Allis-Chalmers, Milwaukee 1, Wisconsin.*

Allis-Chalmers will be glad to recommend spare parts for your equipment — the best protection you have against shut-downs!

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Are Stockpile Requirements Exaggerated?

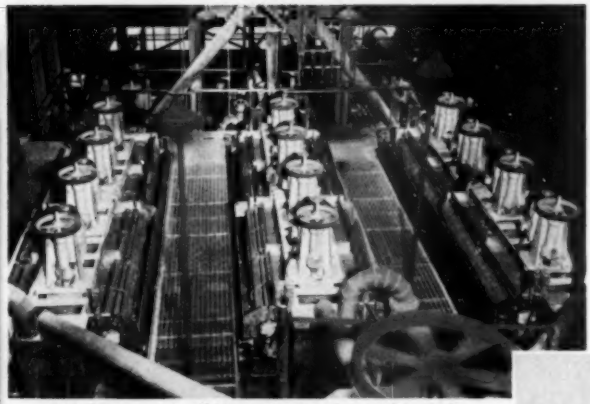
THE federal stockpile of strategic and critical materials was created for an emergency. The emergency has occurred but no withdrawals are being made from the stockpile. On the contrary, the rate of stockpiling is being increased. A serious dislocation between supply and demand for metals exists and purchases for the stockpile are a significant contributing factor in this situation.

Materials can be withdrawn from the stockpile only by Presidential order and apparently this will not be given unless there is a shooting war. Every segment of the economy is feeling the pinch of raw material controls because of the emergency and it is therefore expedient that the methods of disposition and plans for increasing the supplies of critical materials be scrutinized.

Methods of determining stockpile requirements are particularly nebulous to the mine operator and metal consumer. It is known that to determine the amount to be stockpiled, domestic and foreign supplies are estimated on an annual basis for a period of five years, taking into consideration losses for political and military reasons. The sum of civilian and military requirements are similarly estimated. The deficit is the amount which must be stockpiled. The five-year period is selected arbitrarily as the estimated length of hostilities. On the basis of World War II experience, it is not too difficult to estimate civilian requirements. However, military requirements are estimated by the Munitions Board. In the formulation stages of military requirements of a particular commodity, rigid specifications and liberal estimates of amount are likely. How much these original estimates are trimmed before a military requirement figure is arrived at for determining stockpile requirements is not known. Practical considerations should weigh heavily in arriving at this figure at all times, but under present circumstances of restricted use of metals the closest possible estimation for this purpose is imperative. In the case of one rare metal, the military requirements were greater than the known world supplies. This is an extreme case but it is entirely possible that other requirement estimates are similarly out of line with practical considerations. The fact that the amounts of materials stockpiled must for military reasons be kept secret is another example of the extent to which each industry as well as the people are in the hands of a few government administrators.

The Defense Minerals Administration is allocated funds for the development of new sources of supply of raw materials. If these funds could be augmented by the amount that is being spent on stockpiling and a moratorium declared on stockpiling, it is probable that production of many of the nonferrous metals could be boosted to a point where it could satisfy civilian and military consumption two years hence. The objective outlined by C. E. Wilson, defense mobilization administrator, of superimposing production for the military on civilian production might thus be reached in a shorter period of time and with less strain on the economy in the interim. If such a program is not practical, military requirements for the stockpile should at least be re-examined to make sure they are not exaggerated.

ANOTHER CASE OF
RESULTS
WITH FAGERGREN FLOTATION MACHINES!



Typical Fagergren installation at American Cyanamid Co. Sydney mine at Brewster, Florida, floating phosphates.

- Zinc concentrate grade up from 57% to 60%.
- Rougher tailings reduced from 2.5% to 0.6% zinc.
- Horsepower and reagent consumption lowered.

Typical of many installations where Fagergrens are used as both roughers and cleaners, these outstanding results were recently reported by a prominent zinc producer. A conversion from another make machine to Fagergrens provided convincing proof of Fagergren superiority in roughing and cleaning.

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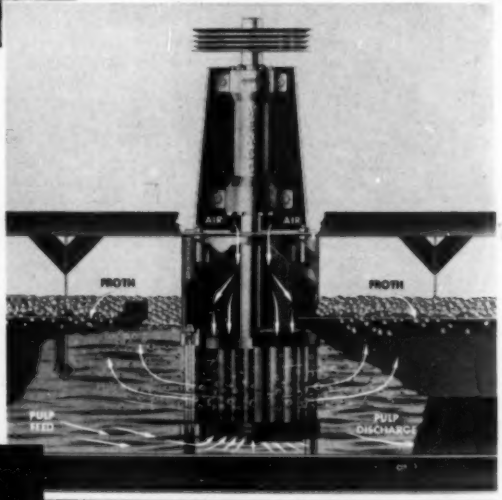

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Trends

A REPORT OF CURRENT ACTIVITY IN THE INDUSTRY

- The Mexican Government is building a mill at Churunibave in Sonora to handle lead, zinc, and copper ores. This development was sparked by increased mining activity in Mexico and Arizona caused by the Korean war.

- Pickands Mather, acting for a group of iron ore producers, is negotiating an agreement with Steep Rock Iron Mines, Ltd. to explore and perhaps lease iron ore property covering more than 1000 acres in the Steep Rock Lake area in Western Ontario. Plans call for the early commencement of an exploratory program. Other members of the group represented by Pickands Mather are Bethlehem Steel, Youngstown Sheet & Tube, the Steel Co. of Canada, and Interlake Iron Corp.

- Cerro De Pasco is the beneficiary of a $20,800,000 credit agreement with the Export-Import Bank of Washington, and will use the money, plus $9 million of its own, to construct a zinc refinery and other facilities near La Oroya, Peru. The installations will have a daily capacity of 200 tons of refined zinc.

- A program for the purchase of tungsten ores, middlings, and low grade tungsten concentrates has been announced by the Pine Creek mine of the United States Vanadium Co., Bishop, Calif. Materials such as pond sludges, slimes, and magnetic rejects are also desired, in view of the strategic position of tungsten. The company is in hopes of treating and converting them into high-grade concentrates.

- One criterion for determining eligibility for government assistance of a proposed project for a new mine or for expansion is its accessibility. Responsibility for certifying the necessity for access roads to develop strategic deposits has been delegated to Secretary of the Interior, Oscar L. Chapman, under the Federal Aid Highway Act of 1950.

- A new alumina plant will be built near Bauxite, Ark., by ALCOA for the purpose of processing low grade bauxite ore. The plant, which will increase by 50 pct the amount of alumina processed by the company, will adjoin the bauxite mining operations of the Alcoa Mining Co.

- A two-year test in the Alabama Power Co.'s Gorgas mine at Gorgas, Ala., has demonstrated that a hot gas suitable for operating a boiler or gas turbine can be produced continuously by burning coal in place. Gas with a heating value of up to 150 Btu was produced for short periods, and the proposed opening of a deeper coal seam may make it possible to produce a gas of 80 to 100 Btu per cu ft without deterioration after a few days operation.

- Scrap metal for the steel industry is being taken from hills of slag which have been accumulating near Coatesville, Pa., for the past 50 years. One of the five slag hills in the locality is estimated to contain 6 million cu yd, and up to 3500 tons of scrap iron have been recovered in one month by a local construction company. A 1½-yd shovel digs into the hill, and the material is transported to a crane which separates the scrap iron from the slag with an electric magnet.

- Industry is mourning the passing of Walter Geist, president of the Allis-Chalmers Mfg. Co., who died on Jan. 29 at the age of 56. Mr. Geist began his career as a messenger in the company's blueprint dept. He had been president for eight and a half years, and at the time of his death was a director of 16 other firms and organizations. Among his honors he counted the Knights Cross, First Class of the Order of St. Olav, presented by Norway's King Haakon VII; honorary doctor of law degrees from Marquette University and Lawrence College; and an honorary doctor of engineering degree conferred by the Michigan College of Mining and Technology.

- A shortage of trained engineers is the most serious bottle neck in the nation's defense program, according to Karl T. Compton, chairman of the board of MIT. Numerous other comments on this shortage have been heard on all sides. AIME is united with ASCE, ASME, AIEE, and AIChE to form the Engineering Manpower Commission, designed to overcome the lack of engineers. Specific programs include: protection of critical personnel in key industries, encouraging college freshmen to study engineering, channeling of engineering graduates into essential jobs, keeping advanced engineering students in college until they graduate providing their work is satisfactory.

- Mining properties in Utah paid 16 pct of the total of $38,168,354 in property taxes collected by that state during 1950. The $6,146,044 paid by the mining industry was further enlarged by taxes paid by blast furnaces and steel mills in Utah County, and smelters at Garfield, Midvale, and Tooele, all of which are classified as manufacturing plants.

- An allegedly new method of extracting platinum from its ore is said to have proved successful in Britain and may result in a new mining operation in Southern Rhodesia. Prospecting rights covering 70 sq mi have been granted in the Belingwe district to a Bulawayo group, and the erection of a pilot plant is said to be under consideration.

- Deposits of iron ore have been located in Pakistan, and a preliminary survey carried out by the Sind Industries Dept. The ore is located at various depths and spread over 30 to 40 mi between Jhole-Dhund a few miles from Thatta, and the 104th mile on the Karachi-Hyderabad road. First samples experimented on in the Punjab laboratories were reported to contain 49 to 56 pct of the metal.

Trends

A REPORT OF CURRENT ACTIVITY IN THE INDUSTRY

AT a meeting of the National Minerals Advisory Council on Jan. 29, it was decided to continue the Council on the original basis as existed before the passage of the Defense Production Act. In other words, the Council will not have a Government representative as chairman and will not have all its business and proceedings subject to strict Governmental supervision as will many industry advisory groups in compliance with the Attorney General's interpretation of the Defense Production Act. However, each member of the Council must now obtain security clearance from the FBI since it is probable that classified material will be submitted to this body for advice.

The Council passed a resolution on renegotiation recommending deletion of the language in Section 106 (a) (3) and (b) of HR 1724 and inserting in place thereof the provisions of Subsections (i) (1) and (i) (3) of Section 701 of the Revenue Act of 1943. This change would mean the complete restoration of the exemption in prior laws from the renegotiation of contracts with defense agencies for the products of a mine, oil or gas well, not processed beyond the first commercially marketable stage. The present so-called exemption—which, in effect, is *not* an exemption—is an attempt to apply the wording of the depletion allowance provisions of the tax laws to the renegotiation of contracts. The Council condemned the exemption provisions of the present bill as "arbitrary and impractical" and "a departure from the tried and proven to the unknown and experimental at a time and under conditions when experimentation is unwise and detrimental to the best interests of the United States."

On the subject of taxation the Council reaffirmed its confidence in its five tax revision recommendations presented to the Secretary of the Interior last August 3 (MINING ENGINEERING, Sept. 1950). Enactment of these recommendations will stimulate exploration and development as much as direct government loans. The Council expressed disappointment that its recommendation of Sept. 1 requesting exemption from excess profits taxes for excess production above the base period was not adequately implemented in the Excess Profits Tax Act.

The Council went on record as favoring a return to some form of the World War II Controlled Materials Plan if the present NPA allocations system fails to provide the necessary metal products needed by mining machinery manufacturers and materials for maintenance, repair, and operation. First, however, the Council felt it desirable to wait and see if present cutback orders in the use of scarce materials and end-use restrictions would have the proper effect.

With regard to exploration, the Council felt that, in the encouragement of exploration, funds contributed by the Government should be matched by private capital so that public funds will not tend to be wasted on frivolous ventures. However, it was recognized that in the case of some of the most strategic metals and minerals the Government may have to depart from the matching principle "in whole or in part." Further, the Government should be entitled to repayment of the amount of its contribution from successful ventures.

As to production incentives, the Council thought the Government should prepare to pay existing higher foreign prices for some metals to insure this country's usual share of those metals. Such a method, it is thought, would meet a deficiency from foreign sources and at the same time tend to lessen the need for domestic production incentives.

As to the establishment of prices in excess of market prices the Council recognized that this problem must be solved separately for each metal or group of metals. In the case of strategic metals which have never been produced under free market conditions in this country "there seems to be no reason why the Government should not pay prices above the market with guarantee of price and market for a fixed period." This could be done either by individual contract or by fixing a guaranteed price for a period. For metals produced in volume domestically, the Council suggested that a flat premium for excess production placed high enough to bring supply and demand into balance would be the most efficient and economical method of solving this problem.

• As expected, much confusion is following the General Ceiling Price Regulation of Jan. 26. The stabilization is on an individual basis instead of an industry basis. The highest price received in the period Dec. 19, 1950 to Jan. 25, 1951 for a product is the freezing point. Thus, some tungsten and manganese producers, who were fulfilling old contracts with the GSA in the base period, were frozen at a price at half the general market price in some cases. On some strategic metals like quicksilver, antimony, and chrome the Office of Price Stabilization declined to state what the price ceiling is or should be. The most unusual aspect in the whole picture is the lack of a ceiling on many agricultural commodities because, according to the law, these products have not reached the parity figure and cannot be frozen. According to an authority in the Defense Minerals Administration that agency has been trying to get the ESA to unfreeze the prices of copper, lead, and zinc as a stimulant to production. If successful, "we could all go home." Contract prices in sales of strategic and critical materials to the GSA pursuant to the Stockpiling Act under contracts outstanding on Jan. 25 or entered into prior to Apr. 1 are not subject to the ceiling price regulation.

• Underground metal miners were added to the list of critical occupations by the Labor Dept. on Jan. 14. This list serves as a guide to the Defense Dept. in considering deferment of reservists and National Guardsmen from active duty. Underground metal miner is defined as one who "performs all or a significant combination of the duties involved in driving underground openings to extract ore or rock." The Selective Service System has not yet established a policy of deferment from the draft for metal miners. The Critical Occupations list may be obtained from the Labor Dept., Room 3218, Washington, D. C.

• At this writing the DMA has established commodity advisory committees on tungsten, zinc, lead, titanium, mercury, antimony, copper, and chrome. The committees on tungsten and chrome have already met and presented their recommendations to the DMA. Tungsten producers told DMA that a guaranteed price of about $50 per short ton unit to run for a period of seven years would bring out all the domestic tungsten needed to satisfy this country's military and essential civilian needs. But DMA wants a two-year limit on the contract. Chrome producers advised a high price, five-year contracts, and provision for access roads to inaccessible chrome-bearing areas. (Incidentally, the Secretary of the Interior has recently acquired authority to certify the necessity for access roads to potential deposits of strategic and critical metals and minerals under the Federal Aid Highway Act of 1950.) The DMA has announced that specific details for carrying out the programs for certain strategic metals such as tungsten, chrome, and manganese will be available to all interested parties shortly.

THE PORTER TUNNEL

will open up 10 million tons of coal for Philadelphia & Reading which can be taken without hoists or pumps.

by E. F. YOUNG

TEN million net tons of marketable white ash coal are expected to be taken within 12 years, without the use of pumping or hoisting equipment, from the Williamstown Basin by the Philadelphia & Reading Coal & Iron Co., which is now developing its 4200-ft Porter tunnel in the area at a cost of two million dollars. The basin, just south of the Big Lick mountain anticlinal, is in the north fork of the southern anthracite coal field, extending from Tower City to the village of Keffers. Veins pitch from 70° to vertical on both north and south dips and extend from the crop of the vein on the mountain top (elevation 1550) to about 1000 ft below sea level. Ten of the fifteen veins in the Basin are minable, ranging in thickness from 3 to 12 ft.

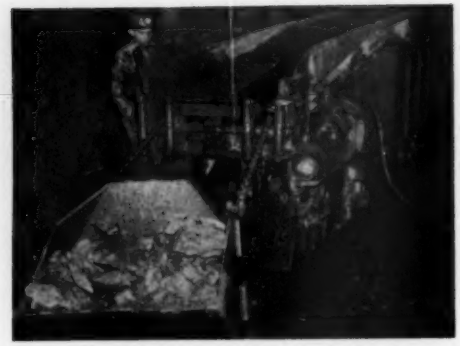
Brookside colliery took 22,500,000 tons of coal from the Basin in 70 years, operating in two veins, and later through three water level tunnels driven in 1917 to mine the top 300 ft of white ash beds. These tunnels were abandoned in 1938, leaving 9000 ft of virgin white ash reserves, estimated to contain 36 million net tons of marketable coal. The Philadelphia & Reading tunnel is being driven into Red Lick mountain at a point directly under the middle tunnel of the abandoned workings. It is designed to cut red ash and white ash measures at an elevation providing a lift of about 300 ft to the overlying workings.

MR. YOUNG is a Mining Engineer with the Philadelphia & Reading Coal & Iron Co., Pottsville, Pa., and an AIME member. This paper was presented before the Pennsylvania Anthracite Section, AIME, Oct. 27, 1950.

Opening the Tunnel

An entrance cut was made into the hillside 357 ft long, 55 ft wide, with an average height of 28 ft in order to provide turnout space for transportation requirements, a ditch into which a 5 ft half-round concrete pipe was laid, and to secure a face of rock strata firm enough for the first sets of timber to be placed at the breaking-in cut or mouth of the tunnel. This excavation, which contained approximately 20,000 cu yd of red, loamy, wet soil, was started July 6, 1948 with company labor and mechanical equipment. Wet weather conditions caused frequent side slope slides and bad transportation conditions, so the open cut was not completed until Sept. 15, when a soft red shale stratum was exposed at the face of the open cut.

Two weeks later the tunnel was started on a +0.5 pct grade, with company labor working three shifts of five men per shift, and driven by hand for a distance of 482 ft. It was driven with an 18-ft width at the rail level, a 14-ft collar and 9 ft high in the clear. A drainage ditch, 5 ft wide at the top,



The Conway loader at work in the Porter tunnel, loading 150 cu ft capacity mine cars.

3 ft wide at the bottom and 3 ft deep, was excavated along the west rib of the tunnel in order to provide drainage for an estimated flow of 20,000 gpm. This section of the tunnel was driven 9x18 ft in cross-section to provide sufficient clearance for concreting or lining the 482 ft of tunnel at some future date.

The soft, shaley nature of the material and the wet conditions in the first 300 ft of tunnel driven obviated firing and all excavation was done with picks and spaders. For the next 182 ft, only light shots were fired to advance the face.

Forepoling and shingling both ribs and face were found necessary before standing a set of timber. Twelve-inch square oak timber was stood on mud sills and securely spragged. This procedure was followed until June 24, 1949 when stratum of a sufficient hardness was reached and, at this point, an independent contractor took over the work of driving the tunnel.

On July 7, 1949, the contractor commenced driving the tunnel which had been reduced in dimensions to a 14-ft spread, 8 ft high clear of the rail, and a 3x5-ft ditch. As the tunnel advanced, the stratum became harder in the red shale measures and all timbering was eliminated excepting where the tunnel cut through the various coal veins. The tunnel is being driven on a continuous, round-the-clock cycle, consisting of four 6-hr shifts: from 7 am to 1 pm is the first muck shift; 1 pm to 7 pm is the first drill shift; 7 pm to 1 am is the second muck shift; and 1 am to 7 am is the second drill shift. Repairing of equipment is done on idle days. A four-man crew, composed of a chargeman, a driller and two helpers, drill and charge a round in from 5 to 6 hr. The drilling is done with two post-mounted drifters.

A 26-hole round is used in advancing the tunnel; the holes average 10 ft and are drilled with hollow steel equipped with detachable bits. Steel in four lengths is used, the maximum length being 13 ft. Holes are charged with 60 pct dynamite in 1½x8-in. cartridges, the normal charge being 200 lb of explosives per round. Charges are stemmed with wood plugs driven into asbestos seals.

Compressed air is supplied at from 85 to 90 psi pressure by two electric air compressors, each having a capacity of 780 cfm. The compressed air is taken into the tunnel through a 6-in. air line to within 500 ft of the face where it is reduced to a 2-in. line to the various working places.

The rock is loaded by a track-mounted Conway shovel which is powered by a 40-hp motor.

A three-man crew, composed of one machine runner, one dresser and one miner, is used for loading the muck and laying the 48-in. gage track.

Cars of 117 cu ft capacity were placed at the mucking machine by a 14-ton combination trolley and storage battery locomotive until July 20, 1949, when 50 all-steel mine cars were put into use. These cars are 150 cu ft water level capacity, equipped with automatic couplers and now are used in both the tunnel and gangways.

The ditch was not excavated as the tunnel advanced due to carrying a 24-in. ventilation pipe along the ditch side. This pipe has now been removed and the ditch is now being advanced in 100-ft sections by three men on the third shift. Holes are drilled with a 55-lb Jackhammer at 4-ft intervals 3 ft deep and fired. The muck is dragged and stocked on a pile with a double-drum hoist and then loaded into mine cars with the mucking machine.

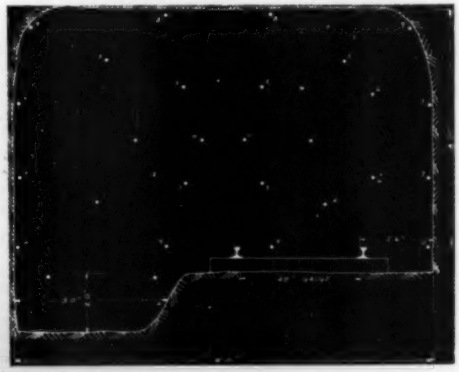
In order to keep the mucking machine supplied with mine cars at all times, a car transfer was placed along the tunnel at various locations where best suited to meet the transportation conditions. After the mine cars are loaded and transported to the outside turnout, an average crew of three men working two shifts with a tractor transport the rock to a spoil bank located about 500 ft from the tunnel mouth. The coal recovered when cutting through the veins was dumped into a bin and trucked to Westwood Colliery.

After the contractor had driven the tunnel 130 ft, it was found that, while the bedding planes were well defined, the measures were confused in spots and there was considerable fracturing of the bedding planes. This fracturing was attributed to the firing in the tunnel and it was not believed that it existed in the measures, except immediately adjacent to the ribs and top of the tunnel. However, mud seams were encountered and as a result the roof presented a ragged appearance and was definitely drummy in spots. No falls of top had occurred, but there was a scaling action which required continual and repeated dressing down.

Roof Bolting

These conditions led to the consideration of a bolting program as a protective measure and, after consultation with representatives of the Bureau of Mines, the plan was adopted. The advancement of

The order of firing of the 26-hole round used in advancing the Porter tunnel.



the face was stopped, the roof was dressed down and the work of pinning was begun. After completing the pinning in 130 ft of the tunnel, it was decided to continue the advancement of the tunnel and complete the pinning in the period of 1½ and 2 hr between the end of the muck shift and the beginning of the drill shift. Advancement of the tunnel continued at the rate of 16 to 17 ft per day while a total of 452 ft of roof was bolted. When hard conglomerate rock was encountered, the pinning was discontinued.

Roof pinning in this particular instance afforded an adequate and efficient method of securing the roof, but, since it did not have the approval of the Dept. of Mines, it was necessary to suspend advancement of the tunnel for 15 days while additional support was installed in the pinned area. This consisted of inserting bolts along the ribs of the tunnel on which 60-lb rails were placed parallel to the ribs with cross-rails laid on the longitudinal rails and planks inserted between the rails and the bolted roof. There is no indication as yet that the auxiliary supports have been subjected to any load by failure of the roof bolts to provide support.

The ventilation for driving the tunnel was provided by a 3-ft reversible fan, equipped with a 7½ hp 440 v ac motor, located at the mouth of the tunnel. This fan delivered 6000 cu ft of air per min through a 24-in. spiral air pipe laid along the west rib of the tunnel. After the tunnel had advanced 1700 ft to the location of the Lykens No. 5 vein, an air hole was driven in that vein for a return airway to the old No. 2 Tower City tunnel which is vertically above Porter tunnel. The fan was then installed in the tunnel at the bottom of the air hole, the air pipe extended to the face, and the fan used as an exhaust for ventilating the development work.

As the tunnel advances, an air hole will be driven in each vein off the ribs of the tunnel for a separate return to the old No. 2 tunnel. Permanent ventilation will be furnished by installing a 50,000 cfm blow fan on a rock airway, now being driven, with the air returning to the surface through overlying old workings.

Purchased power is converted from 66,000 v to 2400 v by a 3000 kva unit-type transformer substation. Power for average motor drive is supplied at 460 v by a 450 kva 2400/460 v transformer station. For development work, haulage power is supplied by a 250 v m-g set.

The water supply is obtained from an 8-in. diam bore hole, 269 ft deep, located near the tunnel. A 50 gpm 247-ft head deep-well pump, operated by a 5-hp motor, delivers the water to a 50,000-gal tank for distribution.

To Feb. 1, 1951, the tunnel has been driven 3789 ft by the contractor in 435 drill-shifts, with an average of 8.7 ft per shift; driven by company 503 ft; making a total of 4292 ft which the tunnel has now been extended into the Big Lick mountain. The driving of the main tunnel has now been stopped for the work of proving the veins on the north dip of the anticlinal.

In order to expedite gangway development, wing tunnels have been driven off the east and west ribs of the tunnel and rock holes are driven to the various veins. The monkey headings are being driven east and west off these rock holes for 60 ft in order to be in a position to connect the ventilation airways to the gangway openings without delay.

Present plans call for main haulage gangways to be driven in the Seven Foot vein for the entire length of the property. Separate turnouts to accommodate 40-car trips will be provided at such locations as necessary to insure efficient transportation. All coal will be delivered to these turnouts by the gathering locomotives and then transported to the cleaner plant by main line locomotives, operating between the cleaner plant and the turnouts.

Off the main haulage gangways, sectional tunnels, approximately 1400 ft long, will be driven at 1200-ft intervals across the basin, cutting both the north and south dips and coal gangways will be driven east and west off these sectional tunnels for a distance of 600 ft. First mining will be done on the advance and, upon completion of the gangways, second mining will start immediately, thus minimizing the expense of relief timbering. Studies are now being made to determine the possibilities of using mechanical mining equipment.

Production from this operation will be rough-cleaned in a cleaner plant approximately 600 ft south of the portal. In this plant, which will have a designed capacity of 500 tons per hour, the large rock will be removed manually from a picking table and the large lumps of coal subjected to crushing in the conventional No. 1 rolls. With the mine in full production, approximately 5000 tons of material will be processed in the cleaner plant during two shifts.

In the early studies of development of this project, an attempt was made to plan for the standard gage railroad service directly to the cleaner plant, but the grades necessary to reach existing railroad tracks were prohibitive. Instead, a 42-in. wide belt conveyor will be installed extending for a distance of 1900 ft on an average grade of 16° to a turnout where the rough-cleaned production will be loaded into railroad cars for transportation to Westwood Colliery or to some other breaker.

A preparation plant is not being planned for this location but the arrangements are such that existing facilities could be incorporated into a preparation plant at a later date.

All outside buildings are of fireproof construction built of cinder block and steel. To date a combination office and storehouse 45x94 ft has been erected; a utility building 32x99 ft, which consists of an ambulance room, first aid room, lamp house, foreman's office and wash room, is also completed. The change room is capable of accommodating 200 men with provision for an extension when the need for additional space is required. A shop, powder house, explosive magazine, compressor house and boiler house are now completed and in use.

In conclusion, the total development progress to Feb. 1, 1951 has been 4292 ft of main tunnel, 828 ft of wing tunnels, 425 ft of rock curves turned and 331 ft of ventilation rock holes and gangways, a total of 5876 ft of rock work completed. Seven gangways are being developed off the main tunnel in order to be in a position to obtain the maximum production as soon as possible after the cleaner plant is erected. A total of 3240 ft of gangway development has been completed to Feb. 1, 1951.

The author wishes to express his appreciation to the representatives of the Philadelphia & Reading Coal & Iron Co. who assisted in the preparation of the paper and to F. E. Kudlich, mining engineer of Glen Alden Coal Co.

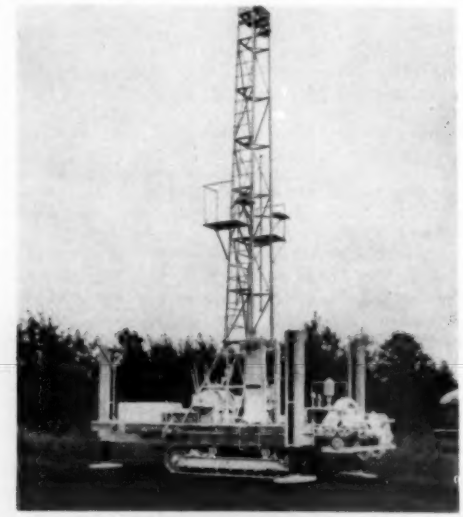
Open-Pit Forum Annual Review

Larger equipment, more attention to maintenance, new types of bits, radio communication, and advances in jet piercing highlight an active year.

by HAROLD N. UNDERHILL, ASSOCIATE EDITOR

OPEN-PIT Mining in 1950 was characterized by even closer attention to the ways and means of reducing production costs. Mines still had their individual problems of terrain, type of rock to be drilled, and the adaptability of specific machines to specific operations, but they were together in their efforts to achieve maximum productivity from a happy marriage of modern equipment and intelligent management.

The South Agnew Mining Co.'s highly successful combination of dragline and belt conveyor, stripping an average of 600,000 cu yd per month, continued during the year, and was also transplanted to the Morton mine near Hibbing. The Morton Ore Co., for which M. A. Hanna is the agent, will strip an oval-shaped area, 1500 ft wide and 4000 ft long, with overburden averaging 180 ft in depth. The South Agnew operation has been using the largest equipment on the iron ranges—a Bucyrus-Erie 1150-B walking dragline with a 30 cu yd bucket does the digging, feeding a huge mobile screening plant which straddles a movable conveyor. The screening plant, a product of the National Iron Co., removes large rocks from the overburden before delivering it to the belt. Five permanent flights now follow the movable conveyor, terminating in a trailing conveyor and a stacker at the dump. Accomplishments at South Agnew represent a high degree of manufacturer and industry cooperation. It is generally believed that the South Agnew operation will set the pace for development of deeper and smaller orebodies to be worked in the future.



Rotary drill, the 58-BH, developed by Joy Manufacturing Co., puts down 6¼-in. holes.

There was an increased interest in maintenance procedures during 1950, as one key to the problem of lowering costs. An interesting study published by the Bureau of Mines dealt with truck haulage at National Lead's ilmenite-magnetite mine in Tahawus, N. Y. There, the trend toward heavier equipment has paid off. In 1949 it cost $0.0925 per long ton to haul ore and waste with a 15-ton truck, and only $0.0743 when a 22-ton truck was used. It was also discovered that hauling costs do not necessarily increase with the age of trucks. It cost $0.1118 a long ton to haul with the four oldest 15-ton trucks in 1948, and $0.1130 per ton with the four newest trucks. It was also discovered at Tahawus that correct balance had to be carefully maintained between primary crusher capacity, number of shovels loading ore, and the number of ore-handling trucks.

From Inspiration, where open-pit mining is done in addition to block-caving operations, H. C. Weed reported (MINING ENGINEERING, August 1950) the establishment of definite maintenance schedules for trucks. These include daily operational maintenance and service, 150-hr service, 450-hr service, plus a series of planned repairs. All items pertaining to Inspiration's trucks are cross indexed on file cards, which makes it easy to locate and note the length of service of any given item of truck equipment. Particular attention is given at Inspiration to tire and tube maintenance, and other operations are following suit. Roads and benches are kept level and clean, and haulage grades kept down to 7 pct.

In drilling, a trend toward the Bucyrus-Erie 42-T churn drills was visible, replacing 27-T and 29-T drills for heavy drilling, except where terrain prohibits the use of the larger drills. One notable exception was the Castle Dome operation, where 29-T rigs were found adaptable to drilling 12-in. holes. Nine-inch holes with wider spacing are becoming popular on the iron ranges. Good fragmentation has been maintained, and drilling and blasting costs reduced. Hole sizes and spacing vary among the open-pit copper mines. Kennecott's operation at

Chino uses 12-in. holes made with a 42-T drill, Cananea operates with a 9-in. bit and 15- to 20-ft spacing, Morenci has had good results with 12-in. holes spaced at 7 to 8 yd, and Inspiration is using 9-in. holes at distances of 18 to 30 ft.

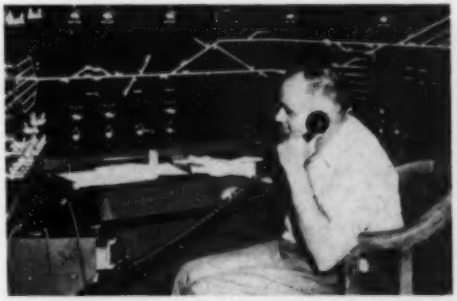
One open-pit copper mine in the Southwest has carried out satisfactory experiments with a tungsten-carbide insert churn drill bit, and Kennametal, Inc., has produced a Samson-type bit which has been used on the Mesabi for structure drilling. Six cemented carbide inserts are set in the cutting edge. The bit is 11½ in. long and sells for $20 to $30, depending on quantity. Rotary drilling methods have been advanced by Joy Manufacturing Co.'s 58-BH rotary blasthole drill, which can put down a 6¼-in. hole to any required depth. It uses compressed air to clear cuttings, and is said to be capable of drilling 15 fph in dense dolomite or 30 fph in ordinary cement limestone. Mobile compressors are also popular, eliminating the need for systems to distribute compressed air to individual drills.

The E. J. Longyear Co. reported an increase in deep-hole drilling on the North American continent during the year. Sectionalized stationary steel derricks, some of greater height permitting longer pulls, are being used more extensively, the company said. Mobile equipment can now be furnished with higher folding masts, such as the Longyear Junior Straitline drill with 20- or 30-ft masts and the Longyear UG Straitline drill with 30- or 40-ft masts.

Although open-pit mining has, on the whole, gone to a truck-and-shovel operation, with conveyors moving in here and there to replace the trucks, the great operation at Bingham Canyon is carrying on with ore trains, and finding it necessary to establish closer coordination throughout the far-flung pit. A mobile radio system has proved particularly effective here since its installation early in 1949. Five base stations are now in service at Bingham—in the mine office, the general mine foreman's pit shack, and at three control points along the trackage. There are six mobile units on electric locomotives, each equipped with a radio transmitter and receiver. At one southwestern copper mine two-way equipment has been placed in every foreman's truck and in tower cars used for trolley work. Radio equipment is also being used in a pit outside the United States, and the use of walkie-talkies is reported from at least one mine.

Various types of auxiliary equipment used in open pits where rail haulage predominates were reported by Charles A. Lindberg of the Oliver Iron Mining Co. (MINING ENGINEERING, March 1950). Three Diesel cranes were delivered to the Oliver operation at the beginning of the year, having capacities up to 55 tons. They feature a new propulsion drive consisting of two 110-hp electric traction motors driven by a 230-v dc generator, which is, in turn, direct-driven by the 225-hp Diesel engine. Some mobile cranes on rubber tires are also in use in rail haul pits, but this use is limited by a lack of roads. Track-laying tractors are also in use for dozer work, hauling, pushing, hoisting, clamming, dragline, skull-cracking and even loading. They range from 4 to 25 tons, with the majority in the 20-ton or over classification. Most popular among the iron range's 190 tractors are the 22-ton diesel-powered machines with 11½-ft straight or 13½-ft angled blades. The trend is toward hydraulic blade control for greater digging power. Special tractors are now being used for track shifting, featuring angle dozers equipped with special lifting hooks. Oliver also makes use of tie tampers—two-stage compressors mounted on rail cars—which supply enough air at 105 cfm capacity to supply eight pneumatic tampers or spike drivers.

A new drilling device for secondary blasting and for getting rid of toes was introduced during 1950, in the form of Linde Air Products' FSJ-1 17-lb hand blowpipe. Burning a mixture of fuel (usually kerosene) and oxygen, the hand blowpipe can drill a 1⅛-in. hole up to a depth of 5 ft, with special models available for deeper drilling. It is especially useful in siliceous rock, which has a high melting point, and thus spalls more readily. To date the blowpipe will not compete with jackhammers in dolomite, diorite, and a few other rocks, but it has been used successfully in sandstone, quartzite and granite.



One of five base stations at Bingham Canyon, at a rail junction point, keeping in touch with nine mobile radio units throughout the pit.

PROGRESS IN DRILLING

tungsten-carbide bits, lighter pneumatic drills, bonus plans, and alloy steels are discussed at northwestern gathering.

THE endless search for cheaper and more efficient drilling methods in stopes, raises, drifts, and headings is still going on in countless underground "laboratories", with results showing up on production sheets and the good news being spread at AIME gatherings. One such gathering was the Columbia Section's program at the recent Northwest Mining Association convention in Spokane, Wash., at which three prominent operators reported on advances in drilling techniques at their respective operations. E. B. Olds, assistant mine foreman at Bunker Hill & Sullivan, described that mine's handling of drill steel and experiments with jackhammers, and he spoke out for the single-use bit which is used almost exclusively in Bunker Hill stopes. Pend Oreille was represented by L. M. Kinney, who was loud in his praise of tungsten-carbide bits, noting that the cost of labor alone for a shorter round with regular steel bits is higher than the cost of carbide bits plus labor for a *longer* round. Longhole drilling at Howe Sound's Holden mine in Washington was discussed by chief engineer W. S. Phillips, who outlined test work with tungsten-carbide bits and drilling problems arising in connection with their use. Some of what these men had to say is herein summarized.

Drilling at Bunker Hill

The vein at the Bunker Hill mine in Kellogg, Idaho, is medium hard, with quartz and quartzite rendering it highly abrasive. Drifter drills are used for drilling with conventional arm and saddle and a 3-in. pneumatic column. Five sizes of single-use bits are available, in 1/16-in. intervals beginning with 1 3/4 in. Bit costs are less than 1c per ft of hole drilled.

Hollow round alloy steel 1 1/8-in. in diam is used, with lugged shanks, a necessary feature because of fractured material in many parts of the vein. Excessive shanking of the rods was an adverse result of the change-over to alloy steel, but this problem was conquered by the addition of a pyrometer to steel shop equipment—a necessary feature in view of the close critical temperature ranges of alloy steel. The steel is heated to 2300°F when it is upset for the lugs, and then cooled for 12 hr in an insulation powder (Sil-o-cel). It is annealed at 1575°F, reheated to the same temperature, and allowed to cool at room temperatures, except for one in. on the lug end, which is subjected to a stream of compressed air for one minute, or until the steel starts to darken. The result is a Rockwell hardness of 42 on the end of the steel, and 35 for the rest of the rod. A grinder in the steel shop sizes, puts a slight taper on, and chamfers the rod in one operation, thus eliminating the necessity for taking the rods to the machine shop.

The drifter setup has proved more amenable to Bunker Hill's type of mining (square setting and filling) than a jackhammer mounted on an air leg, which arragement was given a tryout. A "one-man machine" consisting of an experimental model jackhammer mounted on an aluminum alloy shell, with automatic feed, has been through successful tests. The jackhammer has a 2 5/8-in. bore, is equipped with a steel centralizer, and features a tilting device which permits it to be rolled out of line from the drill hole while drill steel is removed, without changing the drilling alignment of the machine. The machine compares favorably with 3 or 3 1/2-in. drifters when 7/8-in. steel and small bits are used. The small drill holes effect a powder saving, with 7/8-in. powder now in use.

Swedish steel is expected to give domestic steel and bits some competition when results of tests with a Swedish alloy bit, tipped with a tungsten-carbide insert, are finally announced.

Pend Oreille

Timken tungsten-carbide bits and Gardner Denver stopers have been producing 5 to 7 ft of advance per man in each 5x6-ft raise at Pend Oreille. Drilling time is usually 4 hr, with 2 hr for setup time and

Table I

"M" Bits		Carbide Bits	
Cost Labor 2 Shifts	$44.92	Labor 1 Shift	$22.96
" "M" Bits	10.56	Carbide Bits	15.00
	$55.48		$37.96
Cost per ft, 4-ft round	$13.87		
Cost per ft, 5-ft round	7.59		

2 hr allotted for tearing down and loading. The ore rock is a very hard chert with a number of fractures and crevices, and the footwall rock is a soft spongy magnesium limestone. Carbide bits pay off, as shown in Table I, because miners do not have to change bits in a small raise. Length of holes and drilling speed in hard rock have been increased, and the lb of powder per round have been decreased. A hole drilled with 1 3/4-in. carbide bits will take 20 pct less powder than a hole drilled with a regular 2 1/8-in. bit.

A bonus plan is also producing results at Pend Oreille. The old rate of 4 to 5 ft of advance for two miners has been increased to 6 or 7 ft. Two 5x6-ft headings have been substituted for the old 5x12-ft raises. Labor costs have remained constant, powder costs have risen slightly, but the big saving is in timbering and replacement of timber between manway and ore pass. The bonus paid is $2.36 per ft over 5 ft of advance per shift.

Drifting, done with a Joy D. J. T. truck on which are mounted 2 D. J. 8 hydraulic drill jibs and Gardner Denver drifters with 30-in. change and sliding cone, has averaged 8 ft per round, with advances up to 14 ft. Two miners set up and drill out the round in one shift, and an overhead mucker operator and trammer muck out the round, put in ties and rails on the next shift.

It takes 30 min to move the jumbo into the face and prepare for drilling, 5 hr are allotted for drilling, while loading time and putting in pipe requires 2 hr. Ammonia Etra dynamites have replaced the Atlas Giant Gelodyn and du Pont Gelex #2 (with du Pont electric blasting caps) that were formerly used. The new dynamite is a high-cartridge count ammonia-type powder having a slower velocity than gelatin powders, and selling for 75c per hundred cheaper.

Breaking with Longholes at Holden

Since 1945, when ore breaking with longholes began at the Holden mine, numerous equipment changes have been made. In 1947, 1 1/2-in. tungsten-carbide bits were tried in development headings, but the abrasive nature of the rock caused excessive gauge loss. The Holden orebody is a replacement deposit in a highly metamorphised series of sedimentaries, with gneiss and schists in a shear zone several hundred feet wide. Silicification, which took place during mineralization, has given the ore an abrasive drilling characteristic.

Various difficulties with the 1 1/2-in. bit led to trials of the 2-in. bit, which approximated the 1 15/16-in. diamond-drill hole then being drilled. No alternations of pattern, burden, or explosives were necessary, and the 2-in. bit could drive a 55-in. hole with little difficulty, whereas the 1 1/2-in. bit was limited to about 35 ft. Type 2 couplings, 1 1/8-in. round rods, and 3 1/2-in. machines became standard equipment, and great savings in cost per ft of hole were noted as tungsten-carbide bits replaced diamond bits. A big problem arising out of longhole drilling was that of clearing the holes of cuttings, but some experimentation resulted in a successful water swivel and adaptor steel arrangement.

The final step in replacing the diamond drill was the adoption of a 4-in. diam piston machine. This machine, plus 2-in. bits, Type 2 couplings, 1⅛-in. round steel, 1¼-in. lugged shank adaptor steel and the independent water swivel, has become standard equipment.

Adoption of the tungsten-carbide bit has resulted in direct cost savings at Holden. Holes of equal length, up to 70 ft, are now being drilled at less than one third of the cost of the diamond drill method. Other savings are also apparent, among them the reduction in the number of machines from seven or eight down to three or four, using the tungsten bits and 4-in. percussion drill. Less supervision, less maintenance, less lost time and a considerable reduction in inventory and investment round out the picture.

Various other factors entered into the change-over from diamond drills to tungsten-carbide bits. Better techniques had to be worked out for the drill runners. A rigid setup is required to hold holes on line and grade. Three-inch columns with a short arm are standard, and the machine is mounted on a sliding cone shell so that it may be adjusted for required distance from the drill face. Two changes of steel are permitted by the 30-in. change power feeds. To prevent undue strain on the tungsten bits, all holes are collared with 2½-in. conventional steel. For holes collared in the floor of the drift, or other places where cuttings do not drain away, casings are used, with reclaimed 2-in. ID pipe used for the purpose.

Carefully trained men are required to perform longhole drilling, and it has been noted that younger miners with few drilling habit patterns, are more adaptable to the job. Careful and periodic checking of equipment is a must.

Careful bit sharpening is of utmost importance. At Holden, they are sharpened on a pedestal grinder without benefit of a jig. The four cutting edges are ground, but the reaming edges remain untouched. The cutting angle of the inserts is flattened with each successive grind up to 120° in order to preserve as much stock metal as possible.

Bonuses are also in effect at Holden, with 20c per ft paid for each foot drilled over a 50-ft average per bonus shift during the bonus week. It takes the average man about one month to learn enough to start making bonus money.

Table II. Results of Longhole Drilling with Tungsten-Carbide Bits at the Holden Mine

Ft drilled	52,903
Bits used	390
Ft per bit	135.6
Drill hr	3,648
Ft per drill hr	14.5
Ft per new drill steel	181
Ft per adaptor steel	161
Ft per coupling	97
Ft per reground bit	16.7

Longholes are also being used in drift and raise work as well as in stoping. Raise connections up to 60 ft have been made, but the most economical distance is 35 to 40 ft. Holes are loaded from the collar side to the desired depth and blasted with standard electric delay caps. They are then blasted in segments of 6 to 10 ft. Drift and raise work with longhole equipment in good ground has proved economical, especially where raise miners are inexperienced or difficult to obtain. It is believed that the potentialities of longhole drilling have not been adequately explored, and that research will expand them to a point where development headings will be generally advanced with some type of longhole equipment. Table II outlines the results of longhole drilling with tungsten-carbide bits at Holden.

Truck-mounted churn drill.

EXPLORATION FOR

procedures used in the

by HAROLD B. EWOLDT

THE Upper Mississippi Valley zinc-lead district covers an area of some 2000 sq mi located in the southwestern corner of Wisconsin, the northwestern corner of Illinois and a portion of eastern Iowa around the city of Dubuque. Commercial deposits of lead and zinc have been mined in Iowa but the major discoveries during the past eight years have been made in Jo Daviess County, Ill., and La-Fayette County, Wis. Activities in the state of Illinois are centered about the city of Galena, while those in Wisconsin are located a few miles south and west of the town of Shullsburg.

Lead was mined from the region before the Revolutionary War under the jurisdiction of the Spanish, French, and British governments. American interest developed in the district early in the nineteenth century and by 1827 most of the present towns in the area were founded. The end of the Black Hawk War in 1834 marked the beginning of a rapid growth for the area. During the period 1820 to 1865 the district produced some 400,000 short tons of lead to become the largest lead producing area in the world. Figures compiled by the Geological Survey show a production of 767,000 short tons of lead during the period 1798 to 1945 and a production of one million short tons of zinc from 1859 to 1945. In the years 1945 to 1947 the Wisconsin-Illinois district ranked seventh among the zinc producing areas of the United States.

The early production from the district was mainly from the efforts of individuals and partnerships. Shallow deposits and known areas were worked and production gradually declined until 1906 when increased demands for zinc aroused new interest in the region. During the next 20 years many mines were discovered and operated. By 1927 most of the major operators had left the district leaving the field to the Vinegar Hill Zinc Co., which during the past 40 years has profitably mined some 50 orebodies.

In 1942, to meet the increased demand for zinc caused by World War II, the federal government, through the U. S. Bureau of Mines and the USGS, initiated a study of the district. Their work resulted in discovery of the Gray-Bautsch orebody about 5 miles south of Galena, Ill., and now operated by Tri-State Zinc Inc. The success of this discovery focussed attention on the district and the USGS and the Illinois Geological Survey compiled maps of the principal ore producing areas. These maps showed a definite correlation between geologic structure and the location of the ore deposits. Profiting from the information supplied by these maps, Eagle-Picher in 1946 acquired a group of leases north of Galena and made substantial discoveries of zinc ore. Early in 1947, Calumet & Hecla acquired some leases to the south of Shullsburg, Wis., with good results. Vinegar Hill renewed its exploration campaign and encountered good ore deposits adjacent to Calumet's, southwest of Shullsburg.

The area comprising the upper Mississippi Valley zinc-lead district is in the "driftless" or unglaciated area covering the extreme southwestern corner of Wisconsin and the northwestern corner of Illinois. The rocks exposed at the surface belong to the Ordovician and Silurian systems. The formations are flat lying or nearly so, the regional dip being southwesterly, about 25 ft to a mile. This dip of the rocks is not uniform, for the formations gently roll up and down forming structural troughs and crests which meander through the district. The zinc deposits are located along the flanks of these troughs, and one of the first steps in exploration is to determine the character and trend of these structures. Present economic interest in the district is centered in the Platteville, Decorah, and Galena formations of the Ordovician system.

The total thickness of the Ordovician section as exposed in the drill holes is about 430 ft, the Maquoketa shale averaging about 100 ft thick, the Galena dolomite about 220 ft, the Decorah limestone about 30 ft and the Platteville limestone about 80 ft. The zone favorable to the occurrence of large zinc deposits is about 100 ft thick. This zone extends from about the center of the Platteville formation through the Decorah formation and into the lower part of the Galena formation. These formations and their subdivisions have local names supplied by drillers and miners. The stranger in the area will soon appreciate that the Trenton, Glassrock, Oilrock, Blue, Gray and Drab are local terminology applied to various members of the formations comprising the ore zone.

The ore occurs in "pitches and flats." A pitch is a high angle mineralized fracture cutting across the horizontal bedding planes of the country rock, the flats are horizontal mineralized fractures conformable to the bedding planes of the host rock. A series of these pitches and flats containing sphalerite, galena, and pyrite comprise an orebody. The orebodies vary in size, depending on their geologic and geographical locations. An average orebody is about

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LEAD AND ZINC

Illinois-Wisconsin district

AND ROBERT R. REYNOLDS

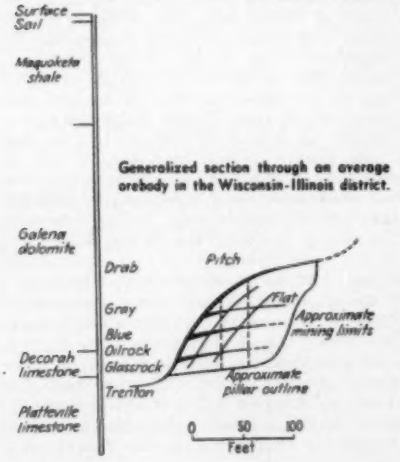
Sampling—flat bottom boiler dumping into 20-gallon tub.

100 ft wide, from 50 to 100 ft high and about 1600 ft long. The orebody may curve and meander in its length. The average orebody will contain about 500,000 tons of ore assaying 6 pct zinc in the form of sphalerite, 0.25 pct lead in the form of galena and about 8 pct iron in the form of pyrite. The grade of ore can be varied by mining methods. In general the smaller the openings and the more selective the mining, the higher the grade of ore. Records are available showing a production from some deposits of ore assaying 15 to 25 pct zinc. The present tendency is to large openings and mechanized mining methods maintaining a 4 to 6 pct zinc ore feed to the mill.

The records of various exploration campaigns for the past forty odd years are available. A resume of the various ore discoveries indicates that the successes fall under three major classifications, called chance discoveries, projection discoveries, and systematic discoveries.

Chance discoveries—This term covers the impromptu ore discoveries made on various occasions by fortunate prospectors while engaged in other pursuits.

Projection discoveries—This method bridges the gap between chance discoveries and systematic discoveries. It presupposes some knowledge of the geology and character of the ore deposits together with the ability to acquire leases on lands adjoining successful mines. As the orebodies tend to follow definite patterns in location, it is a good presumption to prospect along the strike extension of a known ore occurrence. This method has been quite generally accepted and followed by the federal government, and the various companies in the district.



Generalized section through an average orebody in the Wisconsin-Illinois district.

Systematic discoveries—This method, successfully followed to discover the several orebodies now under development, presupposes a knowledge of the character and type of orebodies, a knowledge of the regional geology and the ability to acquire a homogeneous group of leases.

Churn drilling is the accepted prospecting method in the district, offering ease of maneuverability, and a minimum of mechanical supervision. Core drilling has been tried quite successfully. Its major handicaps are lack of a water supply for drilling and upon occasion, excessive bit wear and mechanical difficulties.

A properly coordinated drilling campaign exploring for the ore deposits common to this district serves a dual purpose. First it must discover commercial ore and second it must outline the orebodies both horizontally and vertically in sufficient extent to be used as a guide in the mining operations. The character and the nature of the orebody is such that a vertical drill hole presents the only true picture of grade and continuity.

The drilling program resolves itself into two phases: (1) Scout holes used to determine geologic structures and mineralized areas; and (2) Actual blocking out of the orebody to determine tonnage and grade.

Accurate surveying and logging of the drill holes to rather close tolerances plays an integral part in the successful consummation of the project.

A recommended procedure to be followed during scout drilling is to drill two rows of holes, 660 ft (one eighth mile) apart north and south and 1320 ft (one quarter mile) apart east and west. This spacing of drill holes takes into consideration the legal subdivisions of the land thus reducing crop damage to a minimum. Upon encountering evidence of ore, the drilling pattern should be changed to holes spaced 50 to 100 ft apart, depending upon the apparent width of the orebody, to outline the deposit for thickness, width and length.

Past experience indicates a leased area of some 8000 to 10,000 acres offering an excellent opportunity for new ore discoveries. Close coordination of the leasing, geology, engineering, and drilling programs suggests that about one ton of 60 pct zinc concentrates can be discovered per ft of hole drilled.

Aluminum Antimony Asbestos Amosite Chrysotile Crocidolite Bauxite metal grade refractory grade Beryl Bismuth Cadmium Castor Oil Celestite Chromite metallurgical grade refractory grade Cobalt Coconut Oil Columbite Copper Cordage Fibers Manila Sisal Corundum Diamonds Feathers and Down Fluorspar acid grade metallurgical grade Graphite amorphous lump crystalline flake crucible grade lubricant and packing grade Hyoscine

STOCKPILING

a positive American approach to the problem of "too little and too late"

by JOHN D. MORGAN, JR.

THE tense international situation prevailing in the postwar period, and the outbreak of actual hostilities in Korea, directs attention to the status of United States reserves of strategic and critical materials for war. Geologists and mining and metallurgical engineers are particularly interested in this problem, since, with a few important exceptions, almost all of the materials on the official list of strategic and critical materials for stockpiling (see top of page) are metals and minerals.

Table I. The Status of Selected Major Metals and Minerals in the Pre-World War II Stockpile Program.

Material	Stockpile Objective as of June 1941 (Short Tons)	On Hand in Stockpile as of Jan. 1, 1942 (Short Tons)
Chromite	1,850,000	320,000
Copper	500,000	69,000
Lead	100,000	18,000
Manganese ore	3,696,000	572,000
Tin	271,000	56,000
Tungsten ore	29,000	7,000
Zinc concentrates	150,000	89,000

In the period before the start of World War II there appeared to be little need for concern as to the adequacy of U. S. reserves of most strategic and critical materials. Previous experience and then current estimates of military requirements led to a limited stockpile program that was initiated in 1939 by Public Law 117, 76th Congress. However, the pre-World War II stockpile was concerned almost exclusively with imported materials such as natural rubber and tin. The status of the stockpiles at the start of World War II is shown in Table I, which indicates that objectives were small, and that even these were far from being attained in most cases.

In World War II the United States and its Allies faced a combination of nations which were generally deficient in supplies of most basic raw materials essential to war. Germany, Japan, and Italy had few colonies and but slight raw material wealth within their own borders. Nevertheless, by stringent limitation of civilian consumption and by expert design of military end-products the Axis nations were able to wage war for many years against the Allied nations which controlled most of the raw materials of the world. In the first year of United States entry, sinkings of Allied shipping exceeded new construction, and United States imports of raw materials from Africa and the Western Hemisphere were severely threatened. The United States was forced to uneconomical expansion of domestic sources of raw materials. For example, the synthetic rubber program was necessitated by failure of the Allied nations to stockpile sufficient natural rubber, although the world sources of natural rubber were in the hands of the Allied Nations throughout the prewar period. At the start of the war domestic mine production was accelerated, and maximum rates were achieved in the early war years which, in the face of losses of skilled labor, insufficient supplies of machinery and equipment, and inadequate reserves of developed ore, could not be maintained. United States mine production of copper exceeded one million tons in 1942 and 1943, but by 1945 had fallen below 800,000 tons. United States mine production of lead reached almost 500,000 tons in 1942 but declined steadily to less than 400,000 tons in 1945, while United States mine production of zinc which had reached nearly 770,000 tons in 1942 fell to less than 620,000 tons in 1945. Table II demonstrates the degree to which the United States was forced to rely upon imports in war, although the nation had been virtually self-sufficient in these materials in the preceding period of peace. The significance of imports in meeting needs for all metals and minerals in World War II is shown in Table II. It is to be noted that almost every metal or mineral for which importing to the extent of 25 pct or more was required is now included in the stockpile program.

It became obvious to those responsible for national security planning that a future war fought on a

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Jewel Bearings Instrument Jewels Kyanite Lead Magnesium Manganese Ore battery grade chemical grade metallurgical grade Mercury Mica Muscovite Block Muscovite Film Muscovite Splittings Phlogopite Splittings Molybdenum Monazite Nickel Opium Palm Oil Platinum Group Metals Iridium Platinum Pyrethrum Quartz Crystals Quebracho Quinidine Quinine Rubber Sapphire Ruby Shellac Silk Sperm Oil Talc Tantalite Tin Tungsten Vanadium Zinc

pattern approximating that of World War II would require advance provisions to meet raw material requirements. Accordingly, after extensive hearings, the 79th Congress passed Public Law 520, the Strategic and Critical Materials Stockpiling Act, which was approved by the President on July 23, 1946. The Congress stated that the natural resources of certain raw materials were deficient or insufficiently developed to supply the industrial, military, and naval needs of the country for common defense. The Stockpiling Act established as national policy the acquisition and retention of stocks of raw materials and the encouragement of conservation and development of sources of these materials within the United States in an endeavor to decrease and prevent a dangerous and costly wartime dependence on foreign nations. Public Law 520 specifies that other than for purposes of national defense, stockpiled materials could not be disposed of except for rotation to prevent spoilage or obsolescence. Section 6 of the Act provided for the transfer of government-owned war-surplus stocks of materials to the stockpile in amounts in excess of the needs of industry during the reconversion period. Under this provision substantial quantities were transferred to the stockpile. Section 7 directed the Secretaries of the Interior and Agriculture to undertake supporting actions intended to encourage the conservation of strategic and critical materials, and to expand supplies thereof. In the mineral field much of the regular work of the Bureau of Mines and the Geological Survey has been of material assistance in these endeavors.

Stockpile objectives are determined by comparing estimated wartime supplies of materials to estimated wartime requirements. Stockpiles are intended to cover any deficits that are expected to occur in a five-year total war. In estimating wartime supplies of mineral materials, consideration is given to ore reserves in the United States and in foreign countries; mine, mill, smelter, and refinery capacities; and the possibilities of expansion both within the United States and in strategically accessible foreign countries. In estimating wartime requirements, data are developed as to the military requirements of the United States and its possible Allies, as well as to the needs of war-supporting industries and the minimum essential needs of the civilian population in war. Precise determination of many of these requirements is a difficult problem; fortunately the experience of World War II and the detailed statistics assembled under the Controlled Materials Plan and the Premium Price Plan are of assistance.

As of December 31, 1950, the estimated value of total stockpile objectives was $8.9 billion, and materials worth $2.7 billion were actually on hand. The regular fiscal year 1951 appropriation for stockpiling, as finally approved by the Congress, provided $490 million new obligational authority, including $365 million cash and $125 million contract authority. Since the stockpile is effective storage not only of materials, but also of transportation, manpower, energy, and time—all of which will undoubtedly be in short supply in a future war, the need for acceleration of stockpiling became more pronounced upon the outbreak of war in Korea. Accordingly, at the request of the President, the Congress in late 1950 passed supplemental appropriations which provided $2.4 billion more for stockpiling.

Present indications point to serious shortages of most strategic and critical materials for some time to come. The demands of the accelerated armament program and the stockpiling program superimposed on the normal needs of an expanding national economy require increased supplies of almost all strategic and critical materials. While some of the agricultural materials now in short supply present serious problems, the most significant shortages, and those most difficult to alleviate, are in metals and minerals. The requirements of national security programs have already necessitated credit limitations, inventory controls, and the institution of a system of Defense Orders intended to channel scarce materials into essential programs. Emphasis must be placed on research and development to reduce the need for scarce materials and to develop substitutes.

Table II. The Dependence of the United States on Imports of Metals and Minerals in the War Years 1942, 1943, and 1944.

0% to 25% Imported		25% to 50% Imported	50% to 75% Imported	75% to 100% Imported	
Aluminum	Lithium*	Arsenic	Antimony	Asbestos‡	* Indicates that less than 10% was imported.
Natural Asphalt*	Magnesite*	Copper	Celestite	Beryllium‡	
Barite*	Magnesium*	Ilmenite	Gold	Chromite‡	† The remainder came from secondary recovery.
Bauxite	Scrap and Ground Mica*	Kyanite	Manganese	Cobalt	
Boron*	Molybdenum*	Lead	Sheet and Punch Mica	Columbium‡	‡ Indicates that more than 90% was imported.
Bromine*	Petroleum*	Mercury	Platinum	Corundum‡	
Cadmium*	Phosphate Rock*	Peat	Rutile	Cryolite‡	
Calcium Chloride*	Potash*	Iridium	Silver	Diamond‡	
Cement*	Pyrite	Vanadium	Tin†	Natural Graphite	
Clays*	Salt*	Zinc	Tungsten	Monazite‡	
Coal*	Sand*			Nickel‡	
Emery*	Selenium			Natural Nitrates‡	
Feldspar	Stone*			Palladium	
Fluorspar*	Sulfur*			Osmium, Rhodium, and Ruthenium	
Natural Gas*	Talc, Pyrophyllite, and Soapstone*			Quartz Crystal‡	
Gravel*	Vermiculite*			Radium‡	
Gypsum*				Tantalum‡	
Iron Ore*				Zircon‡	
Lime*					

Air-Powered Saw Speeds Timber Handling

The Wright Air-powered saw is a completely new approach to the problem of power sawing heavy roof and support timbers for mines. Entirely unlike conventional chain or circular saws, the new saw drives twin reciprocating blades over a 4-in. stroke at 1500 strokes per minute. Dynamic balance derived from the opposed motion of the blades eliminates thrust or torque during sawing. The manufacturer, Wright Power Saw & Tool Corp., claims that the saw is as accurate and easy

to handle as a handsaw, but up to 20 times as fast. Weighing 14 lb, it operates from 60 cfm or larger air compressor and is permissible. The depth of cut is not limited and the width of cut is limited only by the length of the 21-in. blades; it can be used for cross-cutting, ripping or notching.

Hartford Accident Insurance, Inc. has covered the manufacturer for accidents caused by design or construction. The sawdust is thrown away from the operator and it cannot bind the blades. The throttle automatically shuts off when the operator releases it. The saw is of aluminum alloy and steel construction. **Circle No. 1**

Belts Now Have Teeth, Numerous Savings Seen

Large scale production of a rubber and fabric belt with teeth has been started by the U.S. Rubber Co. Known as the Gilmer Timing Belt, it is similar in appearance to a flat belt, but features regularly spaced rubber teeth on the inner surface which engage corresponding grooves in the pulleys. It can be manufactured in any desired size. The new belt will not stretch, will operate on fixed centers without take-up adjustments, and it requires no lubrication. The company notes that the new type belt has replaced flat belts, V-belts, chain drives and gears in hundreds of applications. Lowered maintenance costs are said to result from the absence of take-up mechanisms, elimination of lubrication, and elimination of power waste. **Circle No. 2**

Emergency Oxygen Unit

A new demand-type oxygen unit for emergency treatment of smoke inhalation, heart failure, asthma, pneumonia and carbon monoxide inhalation has been developed by Mine Safety Appliances Co. The unit is self-contained in a carrying case, and is operated by opening a cylinder valve and placing the facepiece on the patient. Included are: half-mask facepiece, regulator assembly, 6-ft length of non-kinking breathing hose, 4-cu ft capacity oxygen cylinder, and carrying case. **Circle No. 3**

New 1¼ Cu Yd Payloader

An addition to the PAYLOADER line of tractor-shovels has been announced by the F. G. Hough Co. This one has a 1¼ cu yd bucket, is equipped with 60 hp gasoline or Diesel power. The engine is at the rear over the drive wheels. Automatic bucket tip-back prevents spillage, and hydraulic power control of

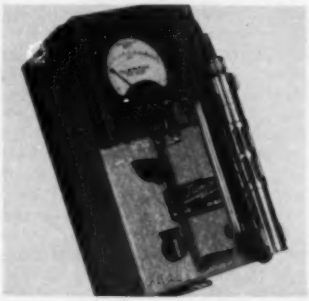
bucket-dump and bucket-close is offered. It will move at four speeds, forward or reverse, with a maximum speed of 29 mph. Digging angle is from 1 to 6°. Maximum dumping clearance is over 8 ft. **Circle No. 4**

New Shaft Seal Said To Eliminate Pump Leakage

A new type shaft seal, the Equiseal stuffing box, has been developed to eliminate leakage on Allis-Chalmers process pumps. It is a radially vaned sleeve which rotates with the shaft to create a pressure which equalizes the suction pressure. As a result, atmospheric pressure exists on the pressure side of the stuffing box as well as outside to eliminate leakage. The Equiseal stuffing box, it is claimed, is not sensitive to wear or close adjustment. It employs no complicated mechanism so that it is able to prevent leakage even when handling viscous liquids or those with solids in suspension. The packing can be removed while the pump is operating on positive suction head without leakage even though there is a direct passage from pump suction to atmosphere. The seal has been designed for use on Allis-Chalmers process pumps with suction heads up to 15 ft at 1750 rpm or six ft at 1150 rpm. **Circle No. 5**

Aid to Uranium Seekers

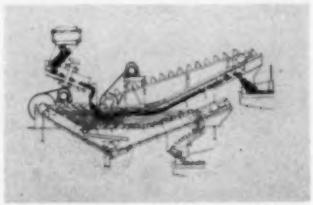
A small, rugged radiation survey instrument, designed to measure radiation intensities (Beta or Gamma) where ac power sources are not available, is being offered by El-



Tronics, Inc. It is a waterproof, fungus-proof unit, and has provision for plug-in phones. It operates from two 1½-v batteries, features Navy-type varicolored scale meter, rotating type Beta ray shield, and wider aluminum case less easily tipped from tilted surfaces. **Circle No. 6**

McNally Acquires Tromp System Rights

Patent and manufacturing rights for the Tromp Heavy Density Coal Cleaning System have been acquired for the Western Hemisphere by the McNally Pittsburg Mfg. Corp., makers of McNally-Norton Baum washers, Rheolaveur coarse and fine coal cleaners, and the McNally Menzies cone. The Tromp system is used in England, France and Holland for cleaning of low-gravity and



high-gravity coals, with or without magnestic media. The first American installation of the Tromp system to process 800 tons per hr is for the Old Ben Coal Corp. mine No. 22 in Illinois. **Circle No. 7**

Free Literature

(8) REFRACTORIES: Reprint of an article describing an integrated chemical-basic refractory plant is offered by *Kaiser Aluminum & Sales, Inc.* Detailed factual information is contained which is useful to consumers of refractories and chemicals in metallurgy and other chemical process industries.

(9) PLASTER AGGREGATE: General brochure on Permalite lightweight plaster aggregate is available from *Great Lakes Carbon Corp.* One of its major uses is fireproofing structural steel, and the entire subject is covered in the brochure. Recommended mixes and applications are listed and a chart lists the materials required for various plaster bases.

(10) AIR MOTOR: Thirty pages of case history photographs, diagrams, wiring circuits, technical data, and dimensional information are contained in bulletin CL-30 issued by *Bellows Co.* The Bellows air motor, an air cylinder with built in valve and speed control regulators, is described and illustrated. Other devices including air-powered drill press feeds, dial feed table, work holding and clamping devices, electronic timing units, standard air cylinders, and standard air control valves are also described.

(11) LUBRICANT: Data book available from *Fiske Brothers Refining Co.* Information is included on control of friction, Lubriplate, various product data and various industries that use this type of lubricant. Information is also included to aid in the selection of the proper grade of Lubriplate for each particular type of equipment.

(12) WIRE ROPE: Catalog 241 offered by *Bethlehem Steel Co.* gives complete information on wire rope and strand. One section is devoted to tables on sizes, weight, constructions, and breaking strengths. Cross-sectional illustrations are shown of the various types of rope manufactured and available.

(13) REAR DUMP TRUCKS: Designed for hauling rock and heavy excavation the *Euclid Road Machinery* truck model FD is described in brochure. Photos show the various types of jobs this truck is capable of handling and specifications are also included as to capacity, dimensions, and all pertinent data.

(14) LUBRICANTS: Lubrication of electric motors and gearmotors is fully discussed in booklet issued by *Socony-Vacuum Oil Co., Inc.* Cross-sectional diagrams illustrate the action of bearings and lubricant in both electric motors and gearmotors. Recommended lubricants are listed and operating notes are given.

(15) RUBBER PRODUCTS: Bulletin BHM-1 available from *Gates Rubber Co.* contains data on air drill hoses, tunnel air hoses, water hoses, sand blast hoses, various v-belts, rubber coverings, and pump diaphragms. Cross-sectional diagrams show the construction of the individual hoses. Data are included on ID, OD, braid, lengths, and working pressure.

(16) AIR COMPRESSORS: The XLE compressor is described in booklet obtainable from *Ingersoll-Rand.* It is a heavy-duty, well-balanced, machine that can be installed with a minimum of effort, time, and cost. This compressor is built for continuous, full-load operation for use in factories, railroad shops, mines, powerhouses, and large construction jobs. Various new designs have been incorporated in its construction such as a new intercooler, cylinder design, and frame and running gear design.

(17) ROTORDRIFTERS: Pamphlet SP-2066 issued by *Chicago Pneumatic Tool Co.* gives complete information on this pneumatic drifter. Available in two sizes of 3 and 3½ in., diam cylinder new features have been added to its construction. It has a short overall length, only one air hose is required, the drill and feed controls are located together on the backhead, and the use of a rotary motor and spur gear drive result in minimum number of working parts and low upkeep of feed parts.

(18) VACUUM FILTERS: Continuous vacuum drum and disc filters for all characters of general dewatering and filtering problems are illustrated in bulletin 4710 from *Morse Bros. Machinery Co.* The 16-page booklet fully describes the various models available, auxiliary vacuum equipment, automatic filter valves, and standard sizes and weight of the drum and disc filters.

(19) HYDRAULIC CONTROLS: Illustrated booklet entitled "Caterpillar Hydraulic Controls" available from *Caterpillar Tractor Co.*, explains in detail how three models of hydraulic controls fit the various models of track-type tractors. Photo cutaways show in detail the construction and operating features of the controls and the hydraulic jacks. Arrangements needed to adapt the No. 44 hydraulic control to either the D2, D4, or D6 tractors are itemized in tabular form.

(20) LIQUID LEVEL GAGES: Instruments for measuring and recording water depth in reservoirs, elevated tanks, rivers, lakes, irrigation systems, process storage tanks, forebay and tail races, flumes, and weirs are described in bulletin L7000 published by the *Bristol Co.* Data is also contained on remote recording and remote automatic pump control of water level with Metameter telemetering instruments. Float pressure, bulb pressure, counterpoise, differential pressure, and air-bubbler types of liquid level gages for a variety of uses are described, including the measurement and automatic control of water and other liquids and solutions, whether flowing, turbulent, corrosive, under static pressure or covered with ice.

(21) SPEED DRIVES: New 12-page bulletin G-509 which describes the basic operating principle of *Reeves Pulley Co.* variable speed drives is now available. Rating tables and dimension drawings are included. Capacities range from ¼ to 87 hp with stepless speed changes within ratios from 2:1 to 16:1. The bulletin covers three basic Reeves units, variable speed transmission, vari-speed motodrive, and vari-speed motor pulley, and a variety of manual and automatic controls for use with them.

Mining Engineering **March**
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Books for Engineers

The Engineering Method. By John Charles Lounsbury Fish. Stanford University Press. 1950. 186 P. $3.—This book is about the "professional habit often referred to . . . . as 'the scientific method' . . ." It formulates for students the system of logic and reasoning that is the mark of the practicing engineer. The three chapters are, "The Method of Engineering", "Application of the Engineering Method" and "Uncertainties in Engineering." They cover logic and reasoning, terms, nontechnical problems, practical conditions controlling the engineering method, inescapable uncertainties and the method of coping with them. The author is emeritus professor of civil engineering at Stanford. The material presented here formed chapters 9, 10, and 11 of *The Engineering Profession*, by Hoover and Fish, second edition 1950.

Elements of Ore Dressing. By Arthur F. Taggart. John Wiley & Sons, Inc., New York, 1950. $10.—A modern and practical introduction to the subject by the noted author of the *Handbook of Mineral Dressing*. The philosophy that a mill and all its parts comprise a single treatment process is a continuing theme in the book. Professor Taggart covers hand picking, screening, filtration, and electrical concentration. After introducing sedimentation, he takes up acceleration in free-settling, hindered-settling, sink-float, jigging, film and trough concentrators and shaking concentrators. A discussion of organic chemicals in flotation is followed by a detailed treatment of flotation. Final chapters deal with primary crushing, secondary crushing, grinding, transport, and controls and flowsheets.

Mechanical Engineers' Handbook. Lionel S. Marks, Editor. McGraw-Hill Book Co. 2236 P. $15.—Written by more than 100 engineers, and covers best currently available information in power generation, aeronautics, welding, metal-cutting, machines, hoisting and conveying, etc. Greatly revised information on such topics as ferrous and nonferrous alloys, super-alloys, silicones, adhesives, powder metallurgy, gas turbines, atomic power is presented.

Geophysical Studies in the Antarctic. Report prepared under Office of Naval Research. By T. C. Poulter. Stanford Research Institute, 1950. 109 P. $4.—This report is of interest to those who contemplate seismic studies of ice fields, whether on shelf ice, continental ice, ice caps, glaciers, permafrost, or ice-and snow-covered land areas. It deals primarily with the geophysical aspects of the Second Byrd Antarctic expedition. Among the sections covered are sound absorption and transmission phenomena, seismic equipment and field procedure, methods of determining depth, physical structure of the shelf ice, snow tremors and geysers, ice movement, surface elevation, tide cracks, and water current.

Similitude in Engineering. By G. Murphy. Ronald Press Co., 1950. 302 P. $7.—Written on the graduate engineering level, this book is an organized presentation of the problems of model design and the interpretation of model tests. In the first part, the concepts and techniques of dimensional analysis are developed and utilized to develop the principles of design of models. In the second, a rational approach to the design of distorted models and results obtained from them are presented. The third and last is devoted to a study of analogies from the standpoint of model-prototype relationships.

AMA Handbook of Wage and Salary Administration, Tested Compensation Methods for Factory, Office and Managerial Personnel. Edited by M. J. Dooher and V. Marquis. American Management Association, New York, 1950. 412 P. $7.50.—Integrating material published by the Association, this manual is a complete survey of the principles and techniques of wage and salary administration and of management's experience in their day-to-day application. It also contains the new AMA research material on wage surveys, union contract provisions covering office salary administration, job evaluation forms, and a bibliography. A model manual used by a company to explain its wage program to employees is reproduced in its entirety as an appendix.

Economics of American Industry. By E. B. Alderfer and H. E. Michl. 2 ed. McGraw-Hill Book Co., New York, 1950. 716 P. $5.50.—Revised and brought up to date, this second edition reflects the technological and structural changes which have taken place since the first edition was published in 1942. Industries included are: metal and metal fabricating, nonmetallic minerals, chemicals, petroleum, rubber, textiles, apparel, food, etc. For each industry, the latest available data are provided to indicate changes in productive capacity, shifts in location, development of new industrial areas, and additional manufacturing facilities.

Economics of Fuel Gas from Coal. By Battelle Memorial Institute; J. F. Foster and R. J. Lund, Editors for Bituminous Coal Research. McGraw-Hill Book Co., New York, 1950. 289 P. $5.—This book contains the results of a survey undertaken by Battelle Memorial Institute for Bituminous Coal Research, Inc. Divided into two main parts, the first deals with the technical aspects of fuel-gas production and includes available data on operating costs, maintenance, and investment cost of gasification processes, and on research. The second part assembles data on the costs of transporting various fuels and discusses supplies, demand, and past prices. From these and other data, future price trends for each fuel are predicted.

How to Chart Timestudy Data. By P. Carroll. McGraw-Hill Book Co., New York, 1950. 323 P. $5.—Of value to industrial engineers and shop executives, this book deals with the fundamentals and methods of charting timestudy data. Following a discussion of incentives and the cost of timestudy, the standard data method is considered. Curve drawing, alignment charts, timestudy equations, multiple curves, preferred numbers, and various methods of charting timestudy data are then considered.

Man the Maker, a History of Technology and Engineering. By R. J. Forbes. Henry Schuman, Inc., New York, 1950. 355 P. $4.—This book explains the development of modern technical achievements as an aspect of the broader history of civilization and culture. It discusses the discoveries and inventions of prehistory, the ancient East, the Greeks and Romans, the Arabs, the Middle Ages, the 16th and 17th centuries, the Industrial Revolution, and the Modern Era. In each case, the close relationship between pure science and engineering and technology is indicated, and the social and economic factors involved in man's conquest of nature are stressed.

Practical Design Considerations for High Tension Belt Conveyor Installations

by J. W. Snavely

THE high tension belt conveyor is introducing a new and tremendously expanded era of low cost bulk material handling. High tension belt conveyors are generally those installations involving very long centers, high lifts, or drops, in which the belts are stressed up to their maximum tension values, and further, where the belt construction provides tension capacity far beyond what is possible with conventional belt constructions. With these high tension installations, the magnitude of the forces involved demands careful refinement of accepted design practice in order to achieve optimum balance of all factors.

No attempt will be made to evaluate the relative merits of belt conveyor haulage with other means of transportation. For present purposes, it is assumed this has already been done in favor of belt conveyor. Neither will any attempt be made to evaluate the various conveyor belt constructions now available or to balance the advantages of various types of mechanical equipment.

It is also assumed that the basic haulage information on which the conveyor design is based is accurate and complete. A sustained maximum, uniform load on the belt at all times must be achieved through proper feed control and the use of adequate surge storage to level the peaks and valleys of any varying demand for the material being handled.

General Belt Capacity Considerations

The belt conveyor capacity tables published by various belting and conveyor equipment manufacturers vary to a considerable degree, and the ratings given are quite conservative. Of necessity, these published ratings are based on the handling of average materials under average conditions.

In applying a high tension belt, all possible capacity from the belt must be obtained in order to hold its width to a minimum and thereby limit the initial cost. Two factors are involved, loading to maximum cross section area and traveling at a maximum practical speed.

Belt Loading: Proper treatment of the loading of the belt will result in maximum cross section to the load, and published capacity ratings can be exceeded, sometimes by appreciable margins. On the 10-mile conveyor haul used in the construction of Shasta Dam, California, although the rated capacity of the belt line was 1100 tons per hr, at times the system handled peak loads of 1400 tons per hr, almost 25 pct better than the rated capacity. One of the large coal companies has been able to exceed rated capacity by as much as 50 pct.

Loading conditions which must be controlled are:

1. Large lumps must be scalped off and rejected or the load must be primary crushed before being placed on the belt.

2. The material weight per cubic foot must be accurate, must be known for all the materials being handled, and must be known for the complete range of conditions of the individual material being handled. Long centers and high lifts magnify small differences into serious proportions.

3. Uniform feeding to the belt is most important. Various types of feeders are available, which can be used to place a constant predetermined volume of material on the belt, or, where an appreciable range of material weight exists, through electrical control actuated by current demand, to place a predetermined uniform tonnage on the belt. One long slope belt in a coal mine in Pennsylvania is being fed at three separate stations with the controls so arranged that whenever the maximum load is going onto the belt from the first station, the other two stations automatically cut out. Whenever the load from the first station drops back, the other two stations again automatically cut in.

4. Careful design of the chutes and skirts is necessary to get the load centered on the belt with a minimum of free margin along each edge. Some free margin at the edge of the belt is necessary to prevent spillage, but if the load can be kept accurately centered, this free margin area can be reduced, and more material can be carried on the belt. What can be accomplished in this respect will vary widely, depending on the nature of the material being hauled. The chute and skirt design must also protect the belt.

5. The design of chutes and skirts should also get the load traveling in the same direction and close to belt speed, so that the load comes to rest on the belt as quickly as possible. The design of the chutes and skirts is worthy of careful study, and after a system is put into operation it should be experimented with to get the best results.

Belt Speed: High belt speeds should be used in high tension work. Obviously, high belt speeds enable haulage on a narrower belt, reducing initial cost.

The major portion of belt wear takes place at the loading point and around the terminal pulleys. The

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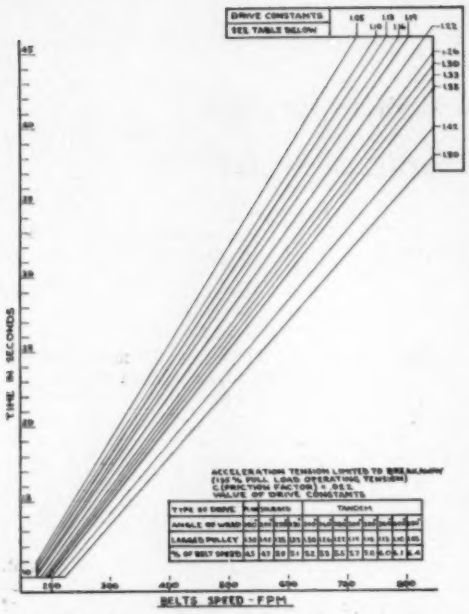


Fig. 1—Minimum acceleration time for high tension belts for horizontal and inclined conveyors.

wear that takes place at the loading point is primarily cover wear, resulting from the scuffing action of the material pouring onto the belt. In addition, there may be damage to the carcass from the impact of large lumps. The wear that takes place at the terminals is primarily to the carcass, from the effect of the flexing around the pulleys. Both types of wear become sharply reduced as the time cycle of the belt lengthens.

The time cycle of a belt can be defined either as the length of time required for any given point of the belt to travel completely around the conveyor, or the number of times per unit of time a given spot on the belt passes under the loading point. Dividing the total length of the belt by the belt speed, or roughly, twice the conveyor centers by the belt speed, gives the time cycle. For example, a conveyor of 1000-ft centers and 500-fpm belt speed would have a time cycle of approximately 4 min.

Long center units permit high belt speeds without seriously shortening the time cycle, and flex life as a limiting factor on belt life can be greatly discounted. Cover wear then determines belt life, and if a given point on the belt passes under the loading point but once every 5 or 6 min, or longer, the life of the cover can be measured in terms of millions of tons handled.

At high belt speeds, the design of transfer becomes most important, both from the standpoint of maintenance cost and material degradation. The degradation of coal at a transfer can be a serious and costly problem. However, in the design of transfers much can be done to minimize the amount of degradation and to make high belt speeds entirely practical.

All of the component parts of the high tension belt conveyor require particular study, the belt, the idlers, the terminal machinery, the drive, and the framework. Even though each of these components must be individually treated, the problems arising from the high belt tensions can be solved only through interdependent component designs. The most important component, because it represents the largest single item of cost, is the conveyor belt.

Selecting the High Tension Belt

The selection of any conveyor belt results from proper balance of strength and quality factors. The high tension belt requires a far more thorough analysis of the strength factors and their determinations than has been necessary for the conventional installation. But as to the quality factors, mainly the covers and rubber used, the application of the high tension belt does not involve any new considerations other than possibly the effect of the loading point time cycle on cover thickness.

Complete analysis of the strength factors involves the calculation and analysis of the belt tensions, with particular emphasis upon the evaluation of horsepower determination formulas, the treatment of slope, and acceleration tensions. The steps involved in determining the tensions for the high tension belt application are the same as for any conventional belt except that all steps must be completely analyzed, particularly for the effect on the total tension. These steps are essentially the following: (1) Horsepower determination, (2) maximum belt tension determination including: (a) effective or working tension, (b) slack side or initial or drive friction tension, (c) slope or weight of belt tension, (d) breakaway or starting tension, (e) acceleration tension.

Horsepower Determination: In the published literature on belt conveyor design, there is agreement in the calculating of the various tensions after the horsepower has been determined, but there is a confusing variety of methods of horsepower determination, with equally confusing results. Inasmuch as high tension applications use substantial amounts of power, it is highly important that the horsepower determination be accurate, and a study of this factor is particularly pertinent.

The power required at the headshaft of any belt conveyor is the algebraic sum of the following four factors: (1) Power to run empty conveyor, (2) power to move load horizontally, (3) power to elevate load, or power generated in lowering load, (4)power to operate trippers.

Only the first and second factors are discussed here. The power needed to lift or lower the load contains no controversial elements in its determination, and the fourth factor will seldom, if ever, be great enough to have differences in its calculation affect the final figure except very slightly. The first two factors evaluate the friction losses of the conveyor with considerable variation of treatment. These differences largely center around two things, the friction factor used to measure the power loss in turning the idler rolls and the measuring of the power loss in the terminal machinery.

Idler Friction Factor: High tension belt applications involve the use of only the highest grade antifriction bearing belt idler equipment. Even with uniformly high-grade equipment, there exists a lack of uniformity in the idler friction factor recommended by different authorities. There is a rather wide variation, some of which is accounted for by the fact that the application of the friction factor is

influenced by several considerations not subject to precise determination. Among such influences are the maintenance of the conveyor including the care of the belt, care of the idlers, cleaning of spillage, the greasing and type of lubricant used, as well as the construction and alignment of the conveyor frame, exposure to the weather, variations in load, spacing of the idlers and diameter of idler rolls. Aware of the effect of these unknowns, from experience the conveyor equipment manufacturers have generally used the conservative 0.03 friction factor. For the average industrial application, the use of this factor has worked out successfully over a long period of time.

Comprehensive studies of friction losses, notably on the 10-mile Shasta line and on one of the earlier long distance coal handling installations, have demonstrated that the 0.03 factor is unnecessarily conservative when all elements of the conveyor design and construction are well carried out. The scatter-band of the readings from Shasta showed that 0.022 was never exceeded.

For the extremely long center belt conveyor it is possible to refine further the use of this friction factor and recognize one more condition which the studies revealed, namely, that the friction losses in moving an empty belt are less than when moving the same belt when loaded. The load produces additional sag of the belt between idlers, and some power is required to lift this load the slight amount over each idler. Further, there is consumption of power by the load in coming to rest on the belt after being discharged onto it and from subsequent changes in the load cross section. The studies made by Hetzel and Albright established a friction factor of 0.015 for moving an empty belt, which is the one used in the following belt conveyor horsepower formulas for maximum tension installations:

(Antifriction bearing equipment exclusive of tripper requirement. Horsepower for tripper equals power consumed by tonnage lift through tripper plus power to propel. See manufacturers' catalogs.)

For average quality installations, industrial and field conveyors,

$C = 0.03 \qquad L_o = 150$

For high quality installations, conveyor transportation lines,

$C = 0.022 \qquad L_o = 200$

$$hp = \underset{\text{(At Headshaft)}}{\frac{C(0.03QS)(L+L_o)}{990}} + \underset{\text{(To Move Empty Belt)}}{\frac{CT(L+L_o)}{990}} + \underset{\text{(To Move Load Horizontally)}}{\frac{CT(L+L_o)}{990}} \pm \underset{\text{(Lift or Drop)}}{\frac{TH}{990}} \qquad [1]$$

For maximum tension installations, slope belt conveyors,

$C = 0.015 \quad C' = 0.022 \quad e = 0.007 \quad h = 0.004$

$$hp = \underset{\text{(At Headshaft)}}{\left[\frac{C(0.03QS)L}{990} + e(0.03QS)\right]} + \underset{\text{(To Move Empty Belt)}}{\left[\frac{C'TL}{990} + hT\right]} \pm \underset{\text{(To Move Load Horizontally)}}{\frac{TH}{990}} \qquad [2]$$

where C = idler friction factor
C' = idler friction factor
Q = weight of moving parts, lb

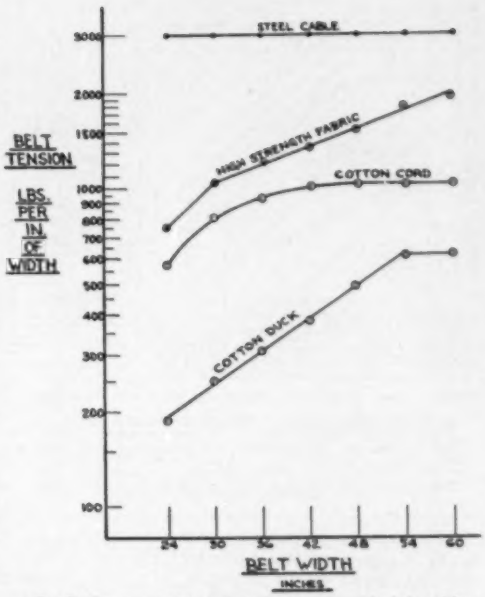


Fig. 2—Conveyor belt high tension rating curves by belt widths and construction.

S = speed of belt, ft per min
L = horizontal centers, ft
L_o = centers correction, ft
T = tons per hr, 2000 lb
H = lift or drop, ft
e = correction factor for added hp, empty belt
h = correction factor for added hp, moving load horizontally

Centers Correction Factor: The power consumed by the belt in going around the terminal pulleys must also be considered. This factor is largely independent of the conveyor centers and its early treatment, while satisfactory for the average length conveyor, overpowered the long center conveyor. Many handbooks of 20 years ago determined terminal friction losses by arbitrarily increasing the power for moving the empty belt and for horizontal conveying by 20 pct for conveyors up to 50 ft in length, 15 pct for 50 to 100 ft, 10 pct for 100 to 150 ft, and 5 pct for conveyors over 150 ft centers.

A more accurate and convenient method was worked out by Goodyear and consists of adding a length constant to the actual horizontal conveyor centers. Graphical analysis of a number of studies showed that with antifriction bearing idler equipment, the terminal consumption of power was about equal to an arbitrary increase in horizontal centers of 200 ft. This 200-ft length constant, or L_o constant by which it is usually designated, is added to the horizontal conveyor centers when calculating the horsepower, as shown in formula 1. The calculated horsepowers using this method are reasonably close to observed operating power demand. In some cases other values have been used for L_o, but considering current experience, this would seem to be unnecessary.

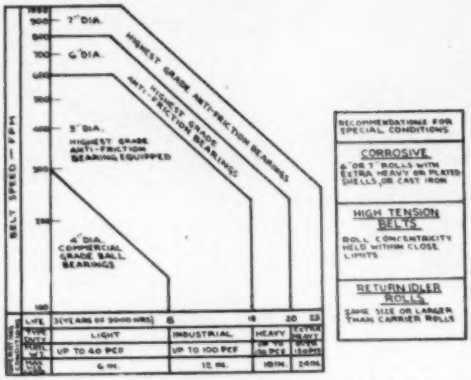


Fig. 3—Selection chart for belt carrier idlers with steel rolls.

Above curves are based on shell life, and correct roll size is shown in area in which it falls. Select correct operating conditions. Use roll diameter shown in this area if belt speed falls within range for chosen area of operating conditions. If belt speed is faster, use roll diameter in area opposite belt speed. If belt speed is slower, use roll diameter in area above operating conditions.

Where maximum stressing of the belt is being applied, it again becomes desirable to refine further the calculation of the terminal factor to provide maximum accuracy. Hetzel and Albright have worked out individual factors to be applied respectively to the empty belt and horizontal conveying formulas, which are reflected in formula 2.

Quality of Installation: It is apparent that difference in treatment exists, but rather than being contradictory, it should be observed that the degree of conservatism used depends on the type of installation involved. Table I shows the different results obtained in calculating the horsepower for the same problem using different current equipment and belting manufacturers' handbooks, together with formulas 1 and 2. At face value, these varying results are confusing, but a reasonable pattern emerges when the logic of the foregoing discussion is applied, and a tabulation of the type of installation for which the formulas were basically intended is superimposed on the results. The thoroughness with which the high tension belt conveyor is engineered eliminates most of the indeterminate factors in industrial and field conveyor installations, and sound engineering consequently dictates the use of the most advanced formula for maximum accuracy and balance of design.

Beyond the treatment of the friction factors, there is little need for further discussion of these formulas, for there are no other differences. It must be emphasized again that the peculiar requirement of the high tension belt is a matter of degree only. The magnitude of the forces involved seriously enlarges what are normally insignificant differences and every possible refinement is necessary to a final balanced design.

Maximum Tension Determination: Having accurately determined the horsepower, the maximum tension for the belt must be established. This is nothing more than a matter of arithmetic, but because of the interrelation of the equipment components, trial computation is required before the best combination can be found. The formulas for determining the slack side or drive friction tension and the other respective tension elements that build up the maximum tension are given in any number of handbooks.

The high tension application requires full consideration of several of the lesser tension demands which, for the ordinary conveyor, have minor importance and can be often ignored. The belt tension demands that must be studied are the slope, acceleration, and breakaway tensions.

Slope Tension: The slope tension found on all inclined or declined units is the weight of the belt hanging from the head pulley, minus the friction of the idler equipment. With slope belts of long inclined centers, the weight of the belt hanging from the head pulley imposes a considerable amount of tension, and it is a factor that must be considered. At the same time, this slope tension must be put to work for it frequently will supply all or most of the necessary slack side or drive friction tension, even for large horsepowers. When it is possible to locate a horizontal gravity type of takeup at the tail end of a slope conveyor, this means that the weight of the belt is supplying practically all of the counterweight tension that otherwise would have to be added.

Acceleration and Breakaway Tensions: Because the high tension belt is highly stressed, the additional tension introduced during the accelerating period when it is started must also be taken into consideration. By predetermining the acceleration time, the additional tension introduced during the starting period can be held to a reasonable figure. With proper electrical control equipment, the acceleration period can be readily set at any desired length of time, 20, 30, or up to 60 or more sec.

The design of the starting controls should provide long enough acceleration time so that the accelera-

Table I. Comparison of Belt Conveyor Horsepower Requirement

Calculated from Formulas 1 and 2 and from Leading Manufacturers' Catalogs

Problem: Required hp at headshaft to convey 1000 tons per hr of coal weighing 50 lb per cu ft, up 15° incline at speed of 660 fpm, 1200 ft inclined centers (1160 ft horizontal centers) and 311-ft lift.

Kind of Installation	Hp to Move Empty Belt	Hp to Move Load Horizontal	Hp to Lift Load	Total Hp at Headshaft
Avg Quality Industrial and Field Conveyors				
Formulas 1 and 2	39.3	39.7	314.1	393.1
Manufacturer A	42.0	41.2	314.1	397.3
B	40.5	39.7	314.1	394.3
C	40.0	38.7	314.1	392.8
High Quality Conveyor Transportation Lines				
Formulas 1 and 2	29.9	30.2	314.1	374.2
Manufacturer D	35.2	37.4	314.1	384.7
E	30.3	30.2	314.1	374.6
Maximum Tension Slope Belt Conveyors				
Formulas 1 and 2	24.3	29.7	314.1	368.1
Manufacturer F	25.5	31.3	314.1	370.9
G	25.8	25.8	314.1	365.7
H	20.9	24.9	314.1	359.9

Table II. Typical High Tension Ratings, Cotton Cord Belts, Lb per in. of Belt Width

No. Plies	GOODRICH No. 50	GOODRICH No. 70	GOODRICH No. 100	GOODYEAR Compass No.		MANHATTAN Ray-Man No.	
4	206			30	200		
5	258	350	457	40	250	33	480
6	309	432	573	50	300	34	600
7	361	512	688			35	720
8	412	592	804	100	450	46	940
9	464	672	918			47	960
10	515	752	1032	125	600	48	1100

tion tension is held at approximately the breakaway tension, that is, the tension required to start the belt and its load from rest and get it moving. Breakaway tension is approximately 130 to 140 pct of the full load effective tension, with the usual design figure being 135 pct. In starting, just enough tension must be supplied to the belt to get it moving, after which the power to get up to full speed must be supplied slowly so that the starting tension is never exceeded. With control of the starting condition, it is not necessary to build as much reserve strength into the belt as would otherwise be necessary, or conversely, for a given strength of belt, maximum effective work can be obtained from it.

It is important therefore that the magnitude of the acceleration forces be accurately determined. It can be shown that the acceleration tension in a belt for horizontal and inclined conveyors varies directly with the belt speed, and inversely with the friction factor used, the type of drive employed, and the time of acceleration. It can further be shown that the acceleration time in seconds will vary within a range of roughly 1/5 to 1/30 of the belt speed in feet per minute. When acceleration tension is held to a breakaway tension of 135 pct, the acceleration time may be obtained quickly and fairly accurately by letting acceleration time in seconds equal 5 pct of the belt speed in feet per minute.

For the operating conditions indicated, an accurate, convenient method of finding the acceleration time in seconds is provided in Fig. 1 in which the acceleration time can be read directly if the belt speed and angle of wrap of the drive are known.

This chart is only for horizontal and inclined conveyors. The problem of a declined conveyor introduces a new condition since the peak operating tension and the peak acceleration tension do not occur at the same point in the belt. A special study of a declined unit is necessary to make sure that the accelerating tension plus the operating tension at the point of maximum accelerating tension does not exceed 135 pct of the maximum operating tension at any point in the system.

Belt Specifications

With the determination of the maximum tension requirements, the type of belt can be selected. In recent years, great strides have been made by the belting manufacturers in increasing the tension capacity of conveyor belts. This development has been along several lines, the more common being the use of cotton cord constructions, high strength fabric construction, and steel cable construction.

No one construction is best for all conditions. A complete study of all the factors is necessary each time, and great variation in the final selection can be expected. For example, with a slope belt problem, although the belting cost might be very much lower if two or three flights are used, this saving in belt cost might easily be outweighed by the cost of providing the necessary machinery rooms underground, plus the cost of drifting a large enough slope to get the heavy drive machinery down to the machinery rooms. In such an instance the use of a very high tension capacity belt, such as a steel cable belt, even though high in initial cost, could easily be the best and most economical selection.

Tables II, III and IV have been compiled from published data of the indicated companies to show what can be expected today from several typical constructions of high tension belts as made by several manufacturers. It can be readily seen from these tables that the horizon for the application of belt conveyors has been pushed back a very great distance.

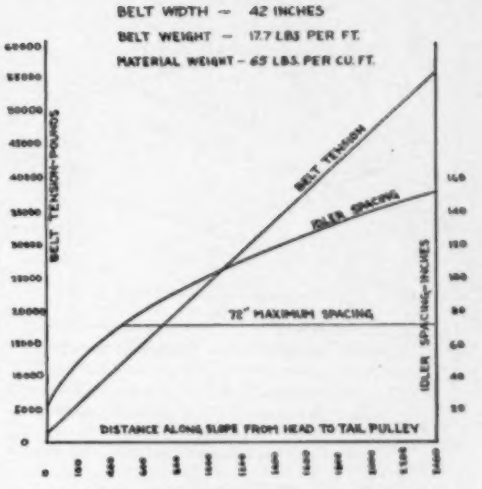

BELT WIDTH — 42 INCHES
BELT WEIGHT — 17.7 LBS PER FT.
MATERIAL WEIGHT — 65 LBS. PER CU. FT.
BELT TENSION
IDLER SPACING
72" MAXIMUM SPACING
BELT TENSION-POUNDS
IDLER SPACING-INCHES
DISTANCE ALONG SLOPE FROM HEAD TO TAIL PULLEY

Fig. 4—Graduated idler spacing curve for high tension belts.

Hauling on the level, one conveyor in Pennsylvania has centers of 2 miles, and it could be possible to carry the 1000 tons per hr of coal in the problems of Table I, on a 42-in. steel cable belt, with 9½ miles between the loading point at the tail pulley and the discharge point at the head pulley.

Along with tension capacity, another important point in belt selection is that of troughability. A conveyor belt must be sufficiently flexible laterally so as to trough properly upon the supporting idlers. This is important both from the standpoint of obtaining the proper load-carrying capacity, and for proper training of the belt to run true both empty and loaded. At the same time, however, a belt cannot be overly flexible because it must also ade-

Table III. Typical High Tension Ratings, High Strength Fabric Belts, Lb per in. of Belt Width

No. Plies	U.S. RUBBER XN	YN	88	CX	CL
4	300	280		440	600
5			425	550	750
6			510	660	900
7			595	770	1050
8			680	880	1200
9			765	990	1350
10			850	1100	1500
11			935	1210	1650
12			1010	1320	1800
13			1100	1430	1950

Table IV. Typical Maximum Tension Ratings, Steel Cable Cord Belts, Lb per in. of Belt Width

GOODYEAR Compass No.		GOODRICH Steel Cord No.		MANHATTAN Whipwire No.	
250	900	1000	1000	17	1000
350	1200	1500	1500	19	1250
450	1500	2000	1800	21	1450
650	2500	2500	2500	26	2000
750	3000	3000	3000	37	2800

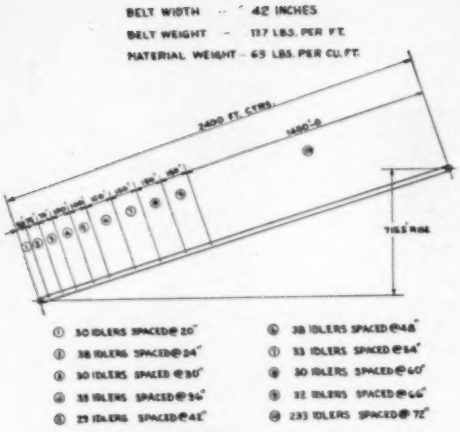

BELT WIDTH - 42 INCHES
BELT WEIGHT - 117 LBS. PER FT.
MATERIAL WEIGHT - 65 LBS. PER CU. FT.
2400 FT. CTRS.
1400'-0
715' RISE
① 10 IDLERS SPACED @ 20'
② 38 IDLERS SPACED @ 24'
③ 30 IDLERS SPACED @ 30'
④ 33 IDLERS SPACED @ 36'
⑤ 23 IDLERS SPACED @ 42'
⑥ 38 IDLERS SPACED @ 48'
⑦ 33 IDLERS SPACED @ 54'
⑧ 30 IDLERS SPACED @ 60'
⑨ 11 IDLERS SPACED @ 66'
⑩ 233 IDLERS SPACED @ 72'

Fig. 5—Graduated idler spacing diagram for high tension belts.

quately support the load it carries; that is, there must not be excessive sag of the load, or dropping of the belt edges between idlers, or creasing of the belt in the gap between idler rolls. With high tension problems, however, because of these tensions, the ability of the belt to support its load is seldom a factor in the belt selection.

Considerable variation in the flexibility of conveyor belts results from the various constructions mentioned. The most flexible is the single plane cord construction, with flexibility decreasing as additional plies of cord or fabric are built up in forming the conveyor belt. The slope of the curves in Fig. 2 is a general indication of the troughability of the respective belt construction, with flexibility increasing as the horizontal is approached.

The lack of troughability has largely limited the tension capacity of conveyor belts in the past. While with standard fabric constructions it is theoretically possible to achieve any desired tension capacity by using the requisite number of plies, troughability has sharply limited what has been practical in the number of plies used. Moreover, it is obvious that the width of the belt also enters in as a factor, which has meant that for the popular widths of 24 to 36 in. more than seven or eight plies could seldom be used. Modern construction of belts has radically changed all that.

Fig. 2 presents the maximum ratings for the constructions shown, and for a given problem, can be helpful in indicating quickly the probable belt construction that will be required. For example, if for a 36-in. belt the maximum tension comes to 2000 lb per in. of width, it is evident that only a steel cable belt can be used. But if the tension requirement is 1000 lb per in. of width, then a high strength fabric belt can also be considered, and since that particular tension falls close to the maximum for cotton cord, this construction also should be carefully checked. Or, if the tension requirement is 300 lb per in. of width, a cotton duck construction will do the job, and the higher cost constructions of belting will not be required.

Fig. 2 is not intended and cannot be used as a basis of selection, but it can provide a quick rough check on recommendations. Assume that a proposal has been received covering a steel cable belt for a belt tension that falls within the cotton cord range. In that case the possibility of using a cotton cord construction, with possible saving in initial cost, should be checked. Ratings falling close to the curves should be particularly checked because of the overlapping of various manufacturers' maximum ratings for approximately comparable constructions.

It must be emphasized that the high tension ratings for conveyor belts that are shown cannot be used indiscriminately but can only be used when every factor of conveyor design is given highest possible quality selection. That means approval from the engineering department of the supplier on the engineering of the conveyor, on the starting controls, and the braking. Vulcanized splices are essential, and pulley sizes must be larger by at least one ply than the handbook recommendation. All of the equipment must be top quality and excellent maintenance must be provided. The belts must use top quality skim coats, and provision must be made for vulcanized repairs. Counterweighted takeups must be provided with ample provision for resplicing the belt. The installation must not be overmotored.

Belt covers and the other quality specifications must be considered. The thickness and the grade of the rubber covers, including such considerations as the tensile strength and friction of the rubber, are little different for a high tension belt than for other conveyor belt applications. The final determination of the belt specifications will depend upon best meeting the individual operating conditions, but without exception, high tension work calls for the very best quality.

One specific point to be mentioned is the effect of the time cycle on cover thickness. High tension conveyor belt applications invariably mean long centers, and this means a long time cycle. Since the major part of cover wear results from the scuffing action at the loading point, a long time cycle often makes it possible to reduce the thickness of the rubber cover. Since this is balancing cover life with carcass life, the overall service life of the belt in terms of millions of tons handled is not affected. Reducing the thickness of the belt cover means saving in the initial cost of the belt.

The long center units involved in high tension work require several splices in the belt because shipping lengths are limited to rolls of belting of approximately 1000 to 1200 ft in length or possibly slightly longer, depending on cover thickness. Only the vulcanized splice can be considered for the high tension belt since it is the only connection that permits utilization of close to the full strength of the belt. The vulcanized splice is 80 to 85 pct efficient, and equally as important, it eliminates the deterioration which is extremely difficult to stop at any mechanical splice.

The splicing of steel cable belts is a matter of considerable interest. One manufacturer uses a method of calculated dispersal of cable ends, so that the load from each discontinued cable is transferred to a group of adjacent cables through rubber in shear. Such a splice can be made to almost any desired strength depending on the extent of the dispersal of cable ends, usually approximately 80 pct of the strength of the belt itself. The cable ends are not directly joined together, but this manufacturer considers the method used, even with its slight loss of

strength, as superior to attempting to join the ends of the cable together, which might result in stiff places in the cable and an unequal tension distribution from cable to cable.

One of the other conveyor belting manufacturers splices its steel cable belt by a means of joining the cable ends together, these joints occurring in a staggered pattern. A special type of connection is used to connect the cable ends, and after a preliminary fastening of the couplings, the entire spliced section of the belt is put under tension so as to obtain a uniform distribution of the tension across the width of the belt. After equalizing the tension, the couplings are permanently fastened to the cable ends. This type of spliced joint also eliminates stiff places in the belt and provides for equal tension distribution from cable to cable.

Idler Selection

The second major item of cost in any high tension or long center belt conveyor application is that of the belt idlers. Ever since the firm establishment of the antifriction bearing type of belt idler, standard idler equipment has been fully equal to the demands of virtually every conveying condition. A wide range of good idler equipment is available today, and for high tension applications, it is only the high grade type of antifriction bearing idler that can be considered. Although all high grade idlers can be used for high tension belt conveyor work, there are some factors that must be considered.

A good seal is required to protect the bearings in applying idlers to high tension work. All high grade idlers have more than ample bearing capacity, and it is highly important that the bearings be kept protected so as to maintain this bearing capacity. For that reason the bearing seal must keep out grit and dirt effectively under all operating conditions. At the same time it must have a long life and not require periodic replacement.

A second important idler selection factor is the necessity of rugged roll construction. In order to have a rugged roll, the wall thickness must be adequate, and construction of the roll must be as a unit. There must be permanent connection of the shell to the ends and of the ends to the bearing tube or roll shaft. No joints should be present to loosen from impact and corrosion. The supporting brackets and the base on which they are mounted must be strong and rugged. If a strong roll construction is not adequately supported, the value of the roll design is largely lost.

In the usual appraisal of idler design, the first consideration is the antifriction bearing used, its capacity and life expectancy. This is only natural since the original replacement of the old plain bearing in idlers was due to the short life and high friction of the plain bearing. Now that the idler industry has had more than 25 years of experience with antifriction bearing idlers, that experience points to a different and more practical approach. Experience shows that roll wear, that is, shell wear, is actually the determining life expectancy factor for high-grade antifriction bearing idlers, where the bearing seals are good and the maintenance is adequate.

Roll Wear: Careful measurement was made of a number of rolls returned from the 10-mile belt conveyor transportation line used at Shasta Dam. These rolls were 5 in. in diam, with a No. 9 gage wall thickness to the shell, or approximately 5/32 in. The belt speed was 550 fpm and approximately 12,500,000 tons of aggregate were carried over a period of

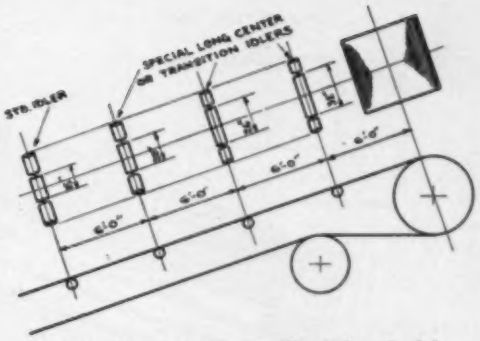


Fig. 6—Use of transition idlers for 42-in. high tension belt.

3½ years, 2 years of which consisted of two 8-hr shifts per day, and 1½ years of one 8-hr shift. The amount of wear that could be detected on the bearings was so small as to be nonexistent, but on the roll shells, the maximum reduction in wall thickness on the rolls was approximately 5/64 in., or about half-way through the wall.

On several other occasions, close examination of bearings taken from idlers that had carried as much as 20 million tons also showed no measurable amount of wear. It becomes apparent that when protected by good seals and adequate maintenance, the life of the bearings in present day, high-grade, antifriction idlers, is not a controlling factor in idler life. Normal idler loads are not great enough to cause measurable bearing wear.

If bearing life in an antifriction bearing idler is virtually indefinite, then it can be ignored as a determining factor in idler life. Instead, it becomes apparent that the roll only must be considered. Further, assuming substantial unit construction, then only the life of the outer shell need be considered because the shell is the only part of the roll subjected to wear.

A number of operating factors affect the roll shell wear, either through the very slight abrading action by the belt in driving the roll, or the effect of shock and pounding upon the roll. Most important and obvious is belt speed, since that determines the number of contacts that the belt makes with the roll. The material load and weight also exert an important influence by determining the degree of pressure with which the belt contacts the idler roll. Where corrosion is present or abrasive dust is in the atmosphere, the wearing action of the belt on the roll is accelerated. Where idlers are tilted slightly forward to help belt training, some sacrifice in roll life is made, resulting from the slight additional skidding action of the belt contacting the roll.

Where large lumps are handled, the shock to the roll often results in deformation, which accelerates shell wear at the high points and correspondingly shortens shell life. However, the additional wear imposed by severe impact and shock is usually secondary since more serious damage results from deforming the supporting brackets, bending the roll shaft, or loosening the roll construction.

The importance of roll diameter to idler life has always been appreciated, but invariably it was the relationship of the rpm imposed upon the bearings by the diameter that was considered significant. With the determining factor of idler life being roll

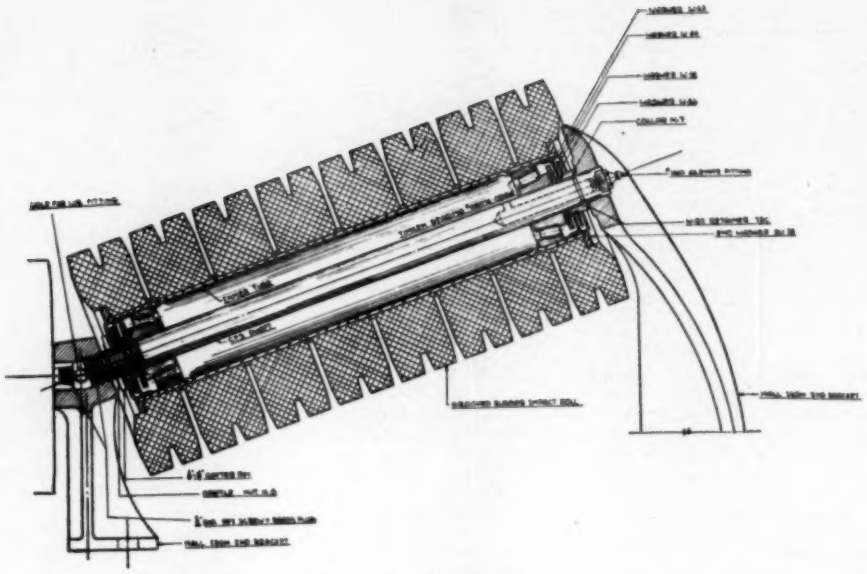


Fig. 7—Impact cushioning idler.

shell wear, the true importance of roll diameter obviously is its determining the number of belt contacts and not the number of bearing revolutions. The larger the roll, the slower the rpm and the fewer number of contacts per unit of time of the belt driving the idler roll. At a belt speed of 500 fpm, the roll rpms for various diameters are: 4 in., 394; 5 in., 318; 6 in., 265; and 7 in., 227. High grade antifriction bearing idlers are available in 5-in., 6-in., and 7-in. roll diameters, and if a 6-in. diam roll is taken as a normal, heavy-duty size, for a given belt speed that means the 4-in. diam roll makes approximately 50 pct more contacts with wear in direct proportion, the 5-in. diam roll approximately 20 pct more contacts and wear, while the 7-in. diam roll has approximately 15 pct fewer contacts and consequently correspondingly less wear. In addition, it is customary for the manufacturers to provide the larger diameter rolls with heavier walls, further penalizing the wear-life of the smaller diameter rolls.

Idler Selection: Since idler life expectancy is a function of roll shell wear, a logical approach to idler selection through the normal range of operating conditions becomes possible. Fig. 3 has been prepared as a guide for the selection of idler equipment for belt conveyor work. It is based upon actual experience of roll wear and the relation between roll diameters and wall thicknesses.

The selection of idlers for belt conveyors has always been a matter of experience rather than any clearly defined rules. One manufacturer made a step forward in simplifying idler selection by applying service factors to the operating conditions in order to determine whether the commercial grade ball bearing idlers should be used or the precision grade, antifriction bearing idlers. Fig. 3 goes further by including roll diameters, and if its use is tempered by experience, it should be a helpful selection tool.

In making idler selections, it has been only too common to give the carrying idler practically all of the attention with little consideration to the return idler. Actually, in all cases, and particularly for the high tension applications, the return idler must be given more attention than the carriers. The return idler rolls are in contact with the dirty side of the belt, so that there is a much more severe abrading action by the belt against the idler roll, resulting in accelerated reduction of the shell wall thickness through wear. It is universal experience that the return idler rolls must be replaced a number of times during the life of a carrier roll.

The return roll construction uses but two bearings located at each end with the roll spanning the complete frame, and deflection must be considered. After a time the belt takes something of a troughed set in its cross section, imposing localized pressure and contact at the roll ends rather than uniformly across the entire face of the return roll, accelerating shell wear at the ends. Moreover, the return idlers carry the belt where it is confined by the frame and difficult to see.

Considering the operating conditions for the return idlers, large diameter rolls should be used to insure maximum life by holding the number of contacts with the belt to a minimum. It is also advisable to provide the return rolls with heavier walls, which will further contribute to longer life.

While any high grade antifriction idler is basically suitable for high tension work, there are some considerations that must be given attention because of the combined effect of high speed and high tension in the belt. One of these factors is roll concentricity.

Roll Concentricity and Materials: For normal belt tensions and speeds, roll concentricity has seldom been a consideration. At high belt speeds, with high tensions, rolls that are seriously out-of-round can

setup and impart very serious vibration. On one recent slope belt installation using a steel cable belt, some of the rolls that were slightly out-of-round, transmitted serious vibration to the steel gallery structure.

For high tension work, it would appear that the concentricity tolerance should be held to a maximum of 0.030 in. total runout on the diameter, particularly when the belt speed is 500 fpm and over. Although this tolerance is practical, it does call for close concentricity limits on the tubing used in fabricating the rolls.

For high tension belt conveyor applications, it is always desirable to provide the return rolls with an extra wide face. More than the usual provision must be made to permit lateral fleeting of the belt without danger of its running off the roll. This is accomplished by setting the frame stringers wider or farther apart, and adding proportionately to the face length of the return roll, usually an extra 6 in.

It is generally accepted practice for belt conveyor installations of all kinds to use either steel or cast iron rolls for the idlers. The high tension belt conveyor does not impose any new roll material requirement. In those instances where corrosion or excessive abrasion is present, the grey iron roll or a coated steel roll is a worthwhile investment.

Special Idler Applications: The spacing of idlers on the conveyor frame is an important cost consideration, since it determines the number of idlers, and therefore initial idler cost. This spacing must be close enough to prevent excessive sag of the belt between idlers, but at the same time, if the spacing is too close, the initial idler cost will be higher than necessary. Tables of average idler spacings based on experience and average conditions appear in the manufacturers' catalogs and handbooks.

In high tension applications, it becomes possible to tailor the spacing of the idlers very closely to belt tension, which simply means that as the tension increases, the idlers are spaced farther and farther apart. Knowing the belt tension, the weight of the loaded belt, and establishing a permissible amount of sag between idlers, the exact spacing of the idlers can be calculated, using the sag formula for a suspended cable. While there is difference of opinion as to the permissible amount of sag, a design figure of a maximum of ¼ in. per ft of span has given good operating performance.

The relationship of idler spacing to belt tension, with permissible sag limited to ¼ in. per ft of span, is shown in Fig. 4. It will be noted that the idler spacing is arbitrarily limited to a ceiling of 72 in., although theoretically a spacing of better than 9 ft is possible. Idler spacings beyond 72 in. have never been used, and as a matter of good practice this idler spacing should not be exceeded.

In designing the conveyor framework, it is more practical to space the idler by zones rather than to use a theoretical spacing for each idler. This obviously provides more economical frame construction, and there is not enough variation between the theoretical and the actual to have any effect upon operating results. Fig. 5 shows how the theoretical idler spacing in Fig. 4 has been converted into ten zones of spacing.

Graduated idler spacing is an important economic consideration since its use saves an appreciable amount of initial idler cost. The number of idlers required in Fig. 5 is 516. Using a conventional spacing of 3 ft 6 in., the number of idlers would be 683.

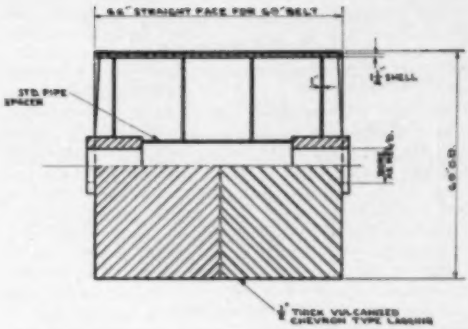


Fig. 8—Typical welded steel drive pulley for high tension belts.

Consequently there is a saving of 167 idlers through the use of graduated idler spacing, or, in this case, approximately 25 pct of the idler cost.

In high tension work it is particularly important that the change in contour of the belt in going from the troughed position on the idlers to the flat position over the head pulley must be done gradually over a considerable distance to keep the stresses in the belt edge from becoming excessive. The normal practice of not supporting the belt between the first idler and the head pulley is impractical because of the unusually long distance. Instead, a gradual change in belt contour is accomplished by means of special transition idlers in which the length of the center roll is gradually increased.

Fig. 6 shows the application of such transition idlers. The cost is nominal since it is possible for the manufacturer to use various standard lengths of rolls and supporting brackets to achieve the combinations indicated.

Although the use of impact cushioning idlers is important for any belt conveyor where lumps are being handled, such idlers are particularly important for high tension work in order to protect the very substantial belt investment. It is the function of the impact cushioning idler to cushion the blow on the belt on the idler roll resulting from lumps dropped on the belt, which it does by absorbing the energy of such impacts through momentary distortion of the roll. Adequate design of an impact cushioning idler must provide sufficient energy absorption capacity over a reasonable range of lump sizes and weights, together with exceptionally heavy and rugged supporting brackets and bases.

Several different designs of impact cushioning idlers are available; one very effective design is shown in Fig. 7.

The subject of training idlers, both carrying and return, is debatable. Theoretically, their use is unnecessary, for a good belt, properly spliced, operating on a well-constructed conveyor frame, can be trained to run perfectly true. Nevertheless, the use of training idlers is frequently desirable for no other reason than to correct unpredictable operating conditions which may develop after starting the system into operation.

Several types of construction of training idlers are available, but as yet none are completely satisfactory, and all of them in varying degrees are apt to cause belt damage. For the return self-aligning units it is particularly important that the belt be

Table V. Minimum Pulley Diameters—Inches

For High Tension Belts

COTTON CORD				STEEL CABLE			
No. of Plies	Drive, Head, or Tripper	Tail Takeup or High Tension Snub	Bend Pulley or Low Tension Snub	Class	Drive, Head, or Tripper	Tail Takeup or High Tension Snub	Bend Pulley or Low Tension Snub
5	42	36	24				
6	48	36	30				
7	54	42	36	1000	54	42	36
8	60	48	36	1500	60	48	36
9	60	54	42	2000	60	48	36
10	72	60	48	2500	60	54	42
				3000	72	60	48

High Strength Fabric

STYLE 85 AND CX					STYLE CL			
No. of Plies	Tandem Drive	Single Drive, Head and Tripper	Tail Take-up High Tension Snub	Low Tension Snub and Bend	Tandem Drive	Single Drive, Head and Tripper	Tail Takeup High Tension Snub	Low Tension Snub and Bend
4	42	30	24	18	42	36	30	24
5	42	36	30	24	48	42	36	24
6	48	42	36	24	54	48	42	30
7	54	48	36	30	60	54	42	30
8	60	54	42	30	60	54	48	36
9	60	54	48	36	72	60	54	36
10	72	60	54	36	72	60	54	42
11	72	60	54	42	84	72	60	42
12	72	72	60	42	84	72	60	42
13	84	72	60	42	84	84	72	48

thoroughly cleaned since any build-up on the return roll is apt to completely defeat the guiding effect.

The construction of training idlers for high tension work must be exceptionally rugged, and generally speaking, standard units are inadequate. In order to obtain a proper steering effect from the training idlers, it may be advisable at times to link two or three such units together in clusters, together with some positive mechanical means of steering the belt.

If the rubber spiral or the rubber disk type of return idler rolls are used to provide additional wear resistance, care must be taken to see that these types of idlers are not located where the return belt is under high tension, as at the head end of slope conveyors where the drive is located at some distance back from the head. The present constructions of such spiral and disk type idlers do not hold concentricity within close enough limits to prevent imparting vibration to the belt.

Terminals

Although the terminal equipment does not represent as much of the total investment as the belt and the idlers, its selection is equally important. Included in the terminal equipment are the head, drive, snub, and tail pulley assemblies, takeup arrangements, as well as such auxiliary devices as belt cleaners.

Pulleys: For drive and high tension snub pulleys, the basic selection factors are those of adequate diameter and strength and grooved vulcanized lagging. The correct relation between pulley diameter and number of plies is given in the handbooks of manufacturers. With belts of multiple ply construction, this relationship is a function of the flex life of the carcass of the belt going around the pulley. With belts where the tension member is a single plane, and particularly with steel cable belts, the factor determining pulley diameter is that of face pressure of the belt against the pulley. Approximately 60 lb per sq in. is the usual maximum allowable face pressure on the pulley.

It is always better to make the pulleys larger in diameter than they need be, rather than smaller, since larger diameter pulleys pay off in better belt operation and longer belt life. But with high tension applications, the head and drive pulleys are of heavy, expensive construction, and consequently there is a cost limitation to making pulleys unnecessarily large in diameter. The various belting manufacturers have all established minimum diameters of pulleys for the several belt constructions which they manufacture, and Table V illustrates several such typical minimum pulley diameter recommendations.

It is most important for high tension belt conveyor applications that the pulleys have great strength with ample factors of safety to withstand the tremendous crushing effect of the conveyor belt, under extremely high tension, wrapped around it. Standard conveyor pulleys, both the welded steel and cast-iron types, are designed for 120 lb per in. of width belt tension, and such pulleys, when furnished as heavy-duty pulleys, have a tension rating of approximately 200 lb per in. of belt width. In high tension work, the belt tensions per inch of width generally start at approximately 500 lb and can be as high as 3000 lb, so it is immediately evident that standard commercial conveyor pulleys are completely inadequate for high tension applications.

The usual design of a high tension pulley is a very heavy, welded steel type of pulley with reinforcing diaphragms. The shell or wrapper is formed from heavy steel plate, as are the diaphragms, with plate thicknesses ranging from ½ in. to as much as 2 in. Such pulleys are completely stress-relieved after welding, and then are machined to size. A typical welded steel drive pulley is shown in Fig. 8. Some very heavy cast-iron, high tension conveyor pulleys have also been used. A welded steel pulley of dead shaft design is shown in Fig. 9. In every case, the pulleys are carefully designed for the specific application.

The type of rubber lagging used on the drive pulley is a grooved lagging of herringbone design. Rubber lagging the drive pulley is a must in high tension work in order to provide the maximum coefficient of friction between the pulley and the belt. Further, it is necessary that this rubber lagging be the grooved type as shown, in order to maintain the high coefficient of friction even if the belt should become wet. Laboratory tests confirmed by field results have demonstrated that the grooved type of lagging has the same coefficient of friction when wet as a smooth lagging when dry, and it is the latter that is used for conveyor design.

Because of the grooving, this type of lagging cannot be applied by bolting as is commonly done with smooth rubber lagging, but, instead, it is vulcanized directly to the conveyor pulley face. The lagging is built up ply by ply right upon the pulley face, and then the whole assembly cured at high temperature for the vulcanizing process. Although the rubber manufacturers have been doing this regularly for the pulley builders, the handling and curing facilities should be checked because of the massiveness of the pulley construction and assembly, which can easily run to 10 tons or more in weight.

In addition to the factors of diameter, strength, and lagging, there is another special consideration

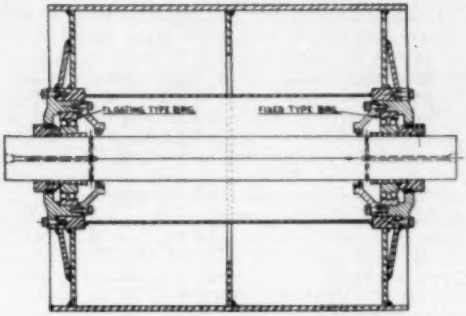


Fig. 9—Welded steel pulley, dead shaft design.

for high tension pulleys. Normally conveyor pulleys are made with a crown, but for high tension belt applications, the pulleys are made with a straight face. Because of the extremely high tensions in the belt and the necessity of distributing this tension uniformly across the entire width of the belt, the use of any crowning is harmful to the belt. This construction has not imposed any operating difficulties, since with high tensions it is easier to keep the belt trained on the straight-face pulley.

One further practical consideration in designing pulleys for high tension work is making the face wider than normal. It has been customary to make the face of the pulley approximately 2 in. wider than the belt for the average size belt, and 3 in. wider for belts of 48 in. and over. However, for high tension applications, the pulley face should be at least 6 in. wider than the belt and occasionally even more becomes desirable. This extra face width is simply to make sure that the belt never runs off the pulley to one side or the other. In spite of everything that is done in training the belt, at times there may be some fleeting on the head pulley, and under conditions of very high tensions, if the edge of the belt actually ran over the edge of the pulley face, serious belt damage could quickly occur.

Where pulleys are located at points of low tension in the belt, standard, heavy-duty design pulleys are used. This always must be checked since it is poor economy to use pulleys without ample factors of safety. There have been many instances where pulleys have had to be replaced with heavier designs after short periods of operation in the field.

Shafting: Shafting for high tension belt conveyor work, involves no special design other than sound, conservative engineering. Because of the larger diameters required, the usual material is forged steel, which occasionally will need to be high carbon or alloy. The design of the shaft is based on combined bending and torsion, and if high carbon or alloy shafts are used, the size must be carefully checked in order to avoid excessive deflection.

Bearings: For high tension belt conveyors, practical considerations dictate that antifriction bearings be used exclusively. With antifriction bearings, the original installation becomes simpler and easier than if sleeve bearings are used, and the customary run-in period of sleeve bearings, with the messy greasing conditions during that run-in period, is entirely avoided. Moreover, the antifriction bearings require only nominal service attention, and they secure the shafts against any lateral float. While the initial cost of the antifriction bearings themselves is higher, there will not be a great deal of difference in the final installed cost, and what cost difference may exist is completely offset by the operating advantages.

Takeup Arrangements: Long center belt conveyors require careful study of the takeup problems, and without exception, some form of a gravity takeup must be used in order to definitely control and provide the correct slack side tension for sufficient driving effort under all conditions. The amount of takeup travel necessary and the location of the takeup pulley in its travel must be considered in addition to providing the required counterweight.

The amount of takeup travel is determined by two considerations, the amount to provide for belt stretch, plus an additional amount to provide for vulcanized splicing of the belt. It is obvious that the amount of travel to provide for belt stretch will be determined by the stretch characteristics of the belt construction used, and the amount necessary can range from almost 2 pct on short center fabric belts to as little as 0.025 pct of the center distance for steel cable belts.

Belt Stretch Allowance: It is obvious that the stretch in a belt will be proportional to the tension in it and that the stretch in the belt is not uniform over its entire length since the tension is not uniform. Consequently, as conveyor centers increase, the amount of takeup movement necessary in percentage of the total centers becomes less. In addition to the effect of tension, temperature and atmospheric conditions also exert influence upon belt stretch.

In addition to providing for belt stretch, a very generous allowance must be made to provide enough slack in the belt to make the vulcanized splice. This is important in order to provide plenty of working length to make the final splice or for any subsequent splicing that may be necessary. The amount of takeup movement to allow for splicing preferably should be sufficient to make two vulcanized splices, or at least 10 ft. An extra 2 or 3 ft will invariably prove to be an excellent investment.

Initial Position of Pulley: The initial position of the takeup pulley in its travel is important for the

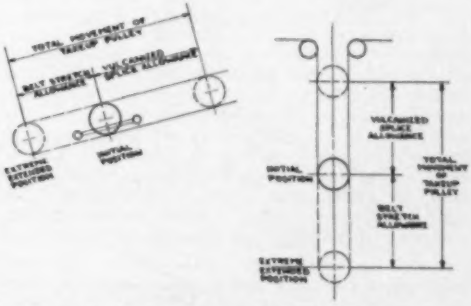


Fig. 10—Takeup movement and initial position takeup pulley for high tension belts.

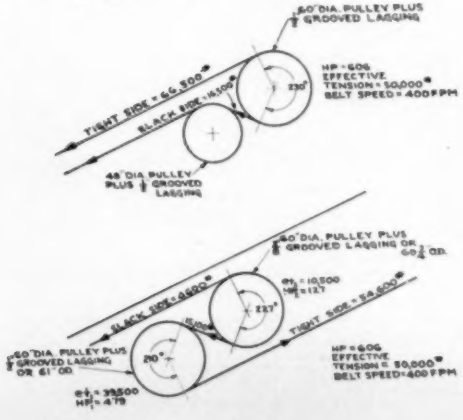
Total movement of takeup pulley, ft = movement back or down from initial position, ft, for belt stretch allowance plus movement forward or up from initial position, ft, for vulcanized splice allowance.
Belt stretch allowance, ft = initial movement allowance, ft, Table VI, X exposure factor, Table VI.
Initial position of takeup pulley = belt stretch allowance, ft, forward or up from extreme extended position.

high tension belt conveyor. Ordinarily the takeup pulley is located somewhere close to the forward position in the takeup so as to allow most of the movement for belt stretch. With a high tension belt application, however, it must not only be possible for the takeup pulley to move back or downward to take care of belt stretch, but at the same time the takeup pulley must be able to move forward or up in order to provide the slack for vulcanizing the belt. The use of Tables VI and VII and Fig. 10 will not only determine the amount of travel to take care of belt stretch under various conditions but will also give the initial position of the takeup pulley. The results from these tables will provide adequate amounts of takeup movement without setting up an excessive amount of movement on conveyors of very long centers.

Gravity Takeup Application: The ideal location for a gravity takeup is at the point of lowest belt tension, for at that point the least amount of counterweight will be necessary. On a horizontal conveyor that would mean placing it immediately behind the drive pulley, on an inclined conveyor at the tail pulley, and on a declined conveyor at the discharge pulley. If the takeup cannot be located at points of low belt tension, then it will be necessary to use a corresponding amount of increased counterweight to make it operative at the point of location.

Fig. 10 shows schematically the difference in action between the horizontal gravity takeup and the vertical type. In the design of vertical units, enough height must be provided for the total movement of the takeup pulley, and it is important that the movement of the takeup pulley be in guides so that there will be no adverse effect upon the training of the belt on its return run. With the horizontal gravity takeup, the carriage should operate on inverted angle tracks which will be self-cleaning, and the wheels as well as the pulley should have antifriction bearings in them. Where great amounts of counterweight are required, dual diameter cable drums or other arrangements of reaving the takeup weight cable may be used to reduce the actual weight used.

Fig. 11—Example of tight side tension for snubbed pulley and dual pulley drives with division of load on dual pulleys.


66,500 / 48-in. belt = 1380 lb tension per in. of width
54,600 / 48-in. belt = 1140 lb tension per in. of width
60" DIA. PULLEY PLUS ½ GROOVED LAGGING
HP = 606
EFFECTIVE TENSION = 50,000#
BELT SPEED = 400 FPM
TIGHT SIDE = 66,500#
SLACK SIDE = 16,500#
230°
48" DIA. PULLEY PLUS ½ GROOVED LAGGING
60" DIA. PULLEY PLUS ½ GROOVED LAGGING OR 60½ O.D.
SLACK SIDE = 6600#
TIGHT SIDE = 56,600#
223°
210°
60" DIA. PULLEY PLUS ½ GROOVED LAGGING OR 61" O.D.
HP = 606
EFFECTIVE TENSION = 50,000#
BELT SPEED = 400 FPM

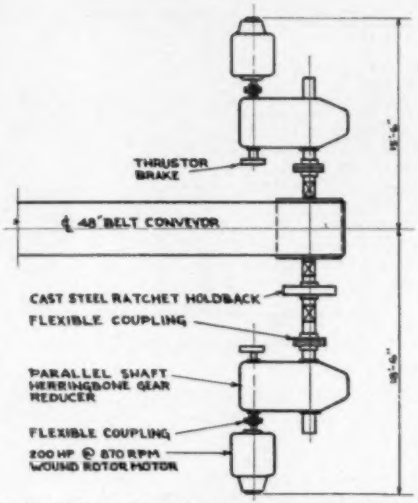


Fig. 12—Snubbed pulley dual motor drive arrangement for high tension belts.

Belt Cleaners: Mention must be made of the belt cleaning problem, because no completely satisfactory cleaning device has yet been built. The common cleaning devices consist of various kinds of scrapers, wipers including spiral wipers, slow and high speed rotary brushes, electric vibrators, and high pressure water sprays. Each has its disadvantages, and the final choice of what to use will depend upon an individual study of the particular cleaning problem.

A belt with material adhering to it can impose serious operating problems on its return run, and in one case, serious study is being made of the possibility of introducing a 180° twist into the conveyor belt at each end of the return run so that the clean side of the belt will be in contact with the return idlers. Where space conditions permit, such a scheme would appear to be practical.

Drives

The tremendously expanded horsepower capacity of the modern, high tension belt now makes drives of hundreds of horsepower a commonplace for belt conveyors, and the record to date is chalked up by an installation now being made which will use a 1500 hp motor. The selection and application of such large horsepower drives for belt conveyors calls for careful study.

Pulley Drives: The drive arrangement for high tension belts is very often made simple by taking full advantage of the high tension capacity of the belt and using a simple snubbed pulley drive arrangement at the head pulley. Applying all of the horsepower to a single snubbed pulley is much preferred to using a dual pulley or tandem drive, which introduces problems of its own. Fig. 11 shows the belt tensions when using a snubbed pulley drive and a dual pulley drive for the same problem. In this case, while the snubbed pulley drive arrangement does impose 12,000 lb additional belt tension, the maximum still falls within the working limits of the belt that would need to be used anyway. The 1140 lb per in. tension in the dual pulley arrangement would require the use of a 1400 lb or 1500 lb

per in. steel cable belt, and the 1380 lb tension for the snubbed pulley arrangement still falls within the limits of that belt. This example is by way of illustration only since in the actual determination of the maximum belt tensions for the final selection of the belt, other considerations enter in. Even if a stronger belt must be used, the additional cost of the belt will usually be less than the added cost of the dual pulley drive arrangement.

One disadvantage of the dual pulley drive is the fact that one of the driving pulleys works against the dirty face of the belt, and particular care must be taken to keep the belt completely clean at all times to prevent the coefficient of friction between the belt and the driving pulley from being seriously lessened. It is possible to introduce additional pulley assemblies to be used as special snub pulleys between the drive pulleys, by means of which both the dual pulleys can be made to drive against the clean side of the belt. Such arrangements add to the initial cost of the installation and considerably complicate the drive arrangement.

When it is impossible to locate the drive at the head pulley, or for good reason all the power cannot be applied to a single pulley, the dual pulley drive arrangement is used very successfully. The design of such drives is fully covered in several handbooks and must be carefully worked out. The normal division of the load on a dual pulley drive is about as shown, although it is perfectly feasible and practical to divide the load between the two pulleys arbitrarily as may be desirable.

It should be noted that the secondary pulley in a dual pulley arrangement should be slightly smaller in diameter than the primary pulley. There are two reasons for this: (1) To keep the tension plane in the belt itself properly centered, inasmuch as there is a difference in the thickness of the carrying cover and the cover on the back of the belt, for example, 3/16 in. as compared to 1/16 in.; and (2) because there is an actual difference in the speed of the belt in going around the two pulleys, with the speed going around the secondary pulley slightly less because of the reduced tension in the belt. A slight difference in the pulley diameters compensates for these two factors.

Table VI. Initial Movement Allowance for High Tension Belt Takeup Travel

Center to Center Distance of Conveyor, Ft	CENTER DISTANCE, PCT			MOVEMENT ON CENTERS, FT		
	Fabric	Nos. 50, 70, 100 Cord	Steel Cord	Fabric	Nos. 50, 70, 100 Cord	Steel Cord
50				4	4	5
150	2.9	2.9		4.50	4.50	5
250	2.8	2.8		7	7	5
375	2.7	2.7		10	10	5
500	2.6	2.6		13	13	5
625	2.5	2.45		16	15	5
750	2.8	2.3		19	17	5
875	2.8	2.05		22	18	5.50
1000	2.5	1.8	0.65	25	18	6.50
1200	2.5	1.8	0.55	30	22	7
1400	2.5	1.8	0.5	35	25	7
1600	2.5	1.8	0.5	40	29	8
1800	2.5	1.7	0.5	45	31	9
2000	2.5	1.5	0.45	0.50	32	9
2200		1.5	0.45		33	10
2400		1.5	0.45		36	11
2600		1.5	0.45		39	12
2800		1.5	0.45		42	13
3000		1.5	0.45		45	13
3200 and over		1.3	0.4		0.013X Center Dist.	0.004X Center Dist.

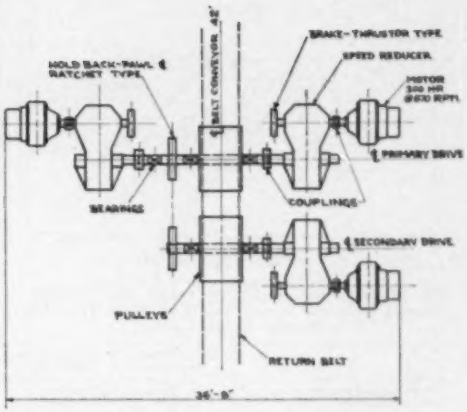


Fig. 13—Dual pulley multiple motor drive arrangement for high tension belts.

Direct Connected Speed Reducer Reductions: While it is entirely possible to use a secondary drive reduction such as a chain drive between the speed reducer unit and the drive shaft, generally it is more practical to couple the slow speed shaft of the speed reducer directly to the pulley drive shaft. The amount of detailed study and consideration that must be given to the high tension belt installation very closely determines the final belt speed, and any future variation in speed is unlikely. The fixed ratio of the speed reducer is not a handicap. Moreover, because of the high horsepower and the relatively slow speed, any secondary drive reduction must be massive in size and often the necessary space is not available. It is important to have the drive reduction as compact as possible, completely enclosed, and operating in oil.

In the large horsepower and the large reduction sizes, the range of selection of standard speed reducers now available is rather narrow with big jumps between sizes. It is therefore frequently necessary to select a reducer unit that is larger in capacity and of higher initial cost in order to obtain one of sufficient capacity.

Motor and Tail Drives: Consequently, it frequently may be possible to achieve initial cost economy through the use of two smaller drives rather than a single large one. While such practice entails care in the selection of the motors and the control equipment, dual and multiple motor drives are entirely practical and such drives are in use. Illustrating what can be done, Fig. 12 shows the use of two motors coupled to a single snubbed pulley drive shaft, while Fig. 13 shows three drives of equal capacity applied to a dual pulley arrangement. With multiple drives, difficulty with any one of them does

Table VII. Exposure Factor for High Tension Belt Takeup Travel

Conveyor	FABRIC BELTS		CORD BELTS	
	28, 32 and 36 Oz.	48 and 42 Oz.	Nos. 50, 70, 100	Steel
Standard or Outside (Covered), pct	80	60	60	40
Outside Not Covered, pct	75	55	55	40
Underground, pct	70	50	50	40

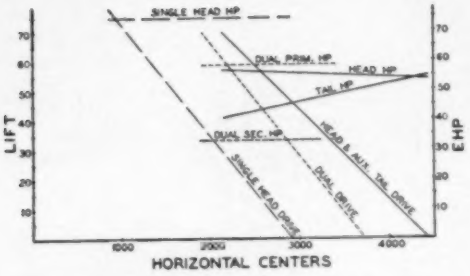


Fig. 14—Comparison of conveyor centers and horsepower. Proposed 5-mile belt conveyor haul, 5 ply 42 oz. 36-in. belt, 350 fpm, 600 tons per hr.

	No. of Flights for 5 Miles	Ft per Flight Horizontal
Single head drive	9	2920
Dual head drive	7	3690
Head and auxiliary tail drive	6	4290

not completely shut down the conveyor, which can continue to operate at reduced capacity. There is also the advantage of economy in the carrying of spares.

In laying out transportation by means of belt conveyor systems, an auxiliary drive can be used at the tail end of individual conveyors in order to convert more of the tension capacity of the belt into working tension. The net effect is to make longer centers possible, with the use of fewer units and therefore a reduction in the number of transfer points. Locating a drive pulley at the tail is simply widely separating a dual pulley drive, moving the secondary pulley to the tail. Doing this eliminates some of the inherent disadvantages of the dual arrangement. A better natural division of load is achieved, the need to slightly vary the pulley diameters disappears, and it is relatively easy to have the clean face of the belt also contacting the tail or secondary drive pulley. The effect of a tail pulley drive on increasing conveyor centers is shown in Fig. 14, which also includes the similar but limited effect of a dual pulley drive. The cost of the greater number of smaller horsepower drives, together with the increased cost of controls and wiring, must be balanced against the advantages of eliminating transfer points, to determine whether or not such an arrangement is economically desirable. Applying a drive at the tail end of a conveyor presents no particular mechanical problem and Fig. 15 shows one possible arrangement.

Motors and Controls: The proper selection of the motors and controls is a problem for the electric equipment manufacturers to solve. Several considerations must be called to the attention of the electrical equipment supplier. Because of close control of the starting torque and the balancing of motor speeds, the wound rotor type of motor is a must, and where dual and multiple motors are involved, they must be carefully matched. On declined units, the winding must be suitable for generation of power by the motor, which thus serves as a brake to govern speed and prevent runaway. Generally it will be more economical to use low-speed motors instead of using larger reductions through the speed reducer units.

With the controls, endless combinations can be worked out, and careful study must be given to the entire control problem. They must provide for delayed or stepped starting, building up blocks of torque for easy breakaway, but must add blocks slow enough during the acceleration period so as not to exceed the amount of torque required for breakaway. With dual or multiple motor arrangements, resistances for balancing motor speeds to the loads must be heavy duty because such motors will always be running on resistance.

With inclined conveyors the controls must include a time delay in the holdback circuit to keep the brake in the set position until the motor has had time to build up sufficient torque to prevent runback. On inclined conveyors, a zero speed switch arrangement is used to set the brake immediately in the event of any mechanical failure in the drive. Where a system of conveyors having interlocked sequence starting is involved the reverse sequence stopping is a must. Provisions for emergency stopping is important. This can be worked out best by running an emergency stop wire or cord the full length of the conveyor enabling emergency stops from any point or by placing a number of emergency stop buttons along the conveyor. A single start station must be used at the head of the conveyor, and an operation procedure must be worked out to prevent restarting until the cause of the emergency stop has been cleared.

Speed Reducers: Except for selecting the most economical size, there is no special problem in applying speed reducers to high tension belt applications. The range of selection in the high horsepower units is narrower than is desirable, and it is hoped that this condition can be corrected by the speed reducer manufacturers. The speed reducers used must be of the highest grade possible and adequately oil lubricated.

The reducer for dual or multiple motor drives must be capable of carrying heavy overloads for short periods of time. In multiple arrangements when one motor cuts out, the remaining drive or drives must then be able to operate overloaded long enough to empty the belt. Conservative selection of the speed reducer equipment is highly important for high tension belt applications.

The couplings that are used must be picked equally conservatively and must be completely dependable. For this heavy duty work the enclosed lubricated gear type of coupling has proved to be most successful, although other types of couplings also have been used.

Holdbacks or Brakes: With inclined or declined belt conveyors the holdback device or brake is an important item of equipment. The runback of a long, loaded inclined conveyor, or the runaway of a declined, loaded conveyor can be very disastrous.

Any of the customary types of holdback devices can be used for high-tension applications, band brakes, ratchet and pawls, roller holdbacks, or thrustor brakes on the motor or reducer shafts. It is of prime importance to see that the holdback device selected is far oversize in capacity. Very conservative selection of the holdback is the cheapest kind of insurance.

Where band brakes are used, the brake lining must be held out of contact during normal rotation, and the high horsepower capacity calls for post brake or hoist design. The slope conveyor of 1500 hp previously mentioned uses a regular mine hoist brake.

Where a ratchet and pawl arrangement is used, the ratchet wheel must be located on the drive

shaft, and the pawl held out of engagement during normal rotation by means of a thrustor. Interruption of current for any reason, stopping, power failure, or by a zero speed switch, immediately drops the pawl.

Conservative practice dictates the use of two holdbacks, a thrustor operated brake on the extended motor shaft or high speed shaft of the speed reducer for all normal stopping requirements, and a mechanical brake on the drive shaft to become operative should there be a mechanical failure of any kind in the drive train.

Declined conveyors introduce problems of their own, inasmuch as there is no holding against reversal of rotation and runback, which immediately rules out the use of any ratchet and pawl or roller holdbacks, and requires the use of some form of band brake arrangement. For declined conveyors, a very generous oversize rating selection with adequate provision for heat dissipation is a must.

Conveyor Frame and Supports: The design of the conveyor frame and its supports for the high tension application does not introduce particular design considerations, although it should be obvious that with the high machinery investment involved, it would be poor economy to skimp on the design of the frame and supports. When located in the open, complete housing for the conveyor must be provided in order to adequately protect the investment in the belt and the machinery.

The frame should be made wider than would be normal, with the distance between the stringers equal to at least the belt width plus 14 to 15 in., and in some cases possibly more. A unitized type of head frame construction in which all mountings for bearings, reducers, and motors are machined after welded assembly, and which insures perfect alignment of the drive elements, is also to be highly desired.

Erection and Operation: The problems of the high tension belt conveyor during the installation of the equipment stem from the massive proportions of the heavy machinery items that must be handled. Careful planning is always necessary, and frequently the problem of transporting the heavy terminal pulley assemblies, the large rolls of conveyor belting, and the heavy pieces of drive machinery, particularly where this must be done under ground, may in itself be the determining factor in dictating the use of a single flight slope conveyor, instead of two or three sections for which a much less expensive belt could be used. The problem of transporting under ground frequently exerts definite limiting effect upon the size of the terminals that can be used.

The problem of reaving the conveyor belt into place and making the necessary vulcanized splices is also one that cannot be treated casually. Inasmuch as several splices are required, adequate time for the splicing must be figured. Including the preparation of the belt ends, which takes far more time than the curing, generally from one to two days' time is required for each splice that is made.

After the belt has been made endless, it must be trained so that it will run true first empty, then full. Even with the most accurate initial setting of all of the idlers, the training period will still require individual adjustment of a large number of the idlers, both carrying and return. Training or aligning idlers should never take the place of proper training of the belt. One operator follows the very

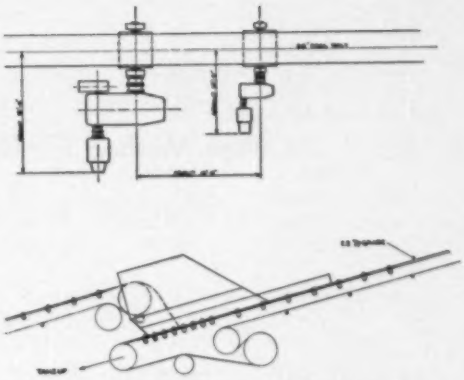


Fig. 15—Auxiliary tail pulley drive arrangement.

sound practice of insisting that his men properly train the belt before he permits the placing of any training idlers in the conveyor.

It is important that maintenance routines be immediately established for proper care of the belt. All cuts and any edge damage must be repaired immediately. While it is seldom possible to shut down long enough to make a complete repair whenever damage is first noticed, it is important to shut down long enough to apply rubber cement to the cut or damage, which will afford protection until the repair can be properly made and vulcanized during an off-shift period. The special cements and adhesives used in repairing belts deteriorate with age, so the supply department must regularly furnish fresh quantities, regardless whether the previous lot has been used or not.

In connection with greasing the idlers, when it is understood that the primary function of grease in a belt idler is not that of lubrication but rather a means of sealing out grit and dirt from the bearing, then it becomes clear that an infrequent greasing cycle is completely adequate. A minimum greasing cycle of approximately 1000 hr of operation is generally conservative, and observation frequently will show that such a cycle can be lengthened. The one exception is operation under extremely wet conditions. Moisture and water tend to emulsify and drain grease from the idler bearing, so that under such operating conditions, frequent renewal of the grease is necessary. Where water conditions are particularly bad, a water repellent type of grease should be used.

The very best evaluation of the recommendations of the belting and equipment suppliers, and the most thorough engineering and cost study of a high tension belt conveyor installation seldom completely covers every last detail that is encountered. Appropriate contingency items, reasonably generous, should be placed in the project budget.

Acknowledgments

Grateful acknowledgment is made of the helpful criticism of S. D. Michaelson, Chief Engineer of Coal Mines, Tennessee Coal Iron and Railroad Co., whose opinions are highly regarded because of his operating experience with one of the outstanding slope belt conveyor installations. Use has also been made of available material, published by conveyor equipment and conveyor belting manufacturers.

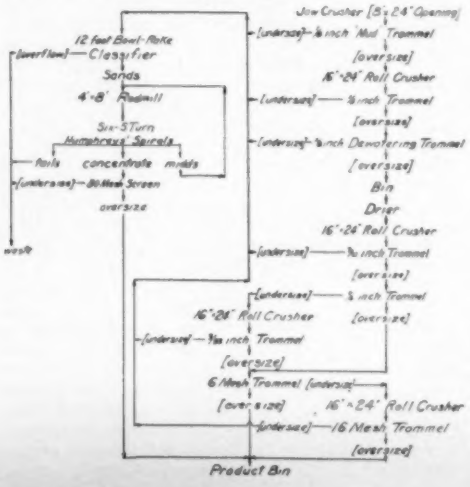
New Method for Recovery of Flake Mica

by R. Adair, W. T. McDaniel, and W. R. Hudspeth

A NEW method for concentrating the flake mica either from present washing plant tailings or from new feed has been developed. In this paper, flake mica refers to that which occurs in weathered granites, alaskites, pegmatites, and schists in a form too fine to be considered sheet mica. It ranges in size from 1 in. or more down to 200-mesh or finer and occurs with kaolin, quartz, and partially kaolinized feldspar. Scrap mica, a term often used indiscriminately to describe everything other than sheet mica, will be used to denote the waste product from sheet mica mines and fabricating plants. Both flake and scrap mica are used in the mica grinding plants to produce wet-ground or dry-ground mica, although some flake mica recovered from clay mining operations is fine enough to be marketed without grinding.

Until 1930, ground mica was made almost entirely from scrap. From 1930 until 1935 a gradually increasing amount of flake mica was mined, and most of this was sold without grinding. A large percentage of this mica was recovered from clay washing plants. In this type of operation the recovery of any sheet mica was of secondary importance and was considered a byproduct.

Fig. 1—Typical mica washing plant revised to treat tailing on Humphreys spirals.



North Carolina produces more flake mica than any other state, accounting for 73 pct of the total domestic production.[1] In the following review of current practice in jig plants, there is no intention to disparage these operations. Within the limitations under which they were built, they are examples of mechanical ingenuity. The term jig plant is a carry-over from early practice since there are no jigs in operation at the present time. Current practice, as shown in Fig. 1, is based on successive stages of differential crushing with rolls followed by trommel screens to remove the more easily crushed gangue minerals. The oversize mica product then is usually dried, and passed through another set of dry rolls and a final screen to remove the remainder of the grit. This process is effective in recovering up to 50 pct of the mica in the original ore, the balance being lost in the screen undersize, usually —⅛ in. Jigs and tables were at one time used to treat this screen undersize and were capable of recovering a part of the mica larger than about 60-mesh.[2] Fine mica lost in these plants is normally washed away in streams and can be considered irrecoverable.

There are ten or more washing plants in the North Carolina area that are losing approximately 50 pct of the mica in the feed. Since the three cooperating agencies, the Asheville Minerals Research Laboratory of North Carolina State College, the North Carolina Dept. of Conservation and Development, and the Tennessee Valley Authority, are concerned with conservation of minerals, they have investigated means for improving the efficiency of these mica plants. The mica industry has been cooperating fully with the laboratory to establish a more efficient method of recovery for flake mica.

Choice of Methods

The initial investigation was concentrated on finding a means of reducing the losses from plants now using the conventional flowsheet shown in Fig. 1. To accomplish this it would be necessary to collect the screen undersize products through the plant and remove from them the mica from ⅛ in. down to

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150-mesh or finer. Since the size of any of these operations is small and capital limited, the method had to be one which could be set up at moderate cost and operated by the present labor force.

Flotation, tabling, and jigging were considered, but before starting work on these, a test was run on a Humphreys spiral, which gave evidence of a definite separation of the mica from the gangue minerals. As compared to other gravity methods or flotation it appeared to offer the following advantages: (1) lower capital cost than flotation or jigs and tables; (2) less skilled labor required; (3) lower operating cost.

Since the ore is relatively cheap and abundant, recovery is not of first importance. Later tests proved that flotation would give the best metallurgical results. The spiral was rated next in efficiency followed by jigs and tables.

Concentration By Humphreys Spiral

This separation is based on shape difference, whereby the thin flakes are carried to the outside of the stream and the more compact particles of gangue are drawn down the inner ports. Mica would move toward the ports were it not for the shape difference, since its specific gravity is slightly higher than that of the gangue.

The important variables tested were feed rate, percent solids in feed, amount and distribution of wash water, degree of grinding of the feed, and extent of middlings recirculation. Analysis of the samples taken from spiral tests was made by separating them by flotation into two or more products whose mica content could be accurately appraised by examination with a binocular microscope. For example, a sample containing 50 pct mica would be very difficult to appraise, but it would be relatively easy to estimate the gangue in a +95 pct mica concentrate or the mica in a clean tailing. Some fine fractions were analysed by means of a heavy-liquid separation in a high-speed centrifuge. The slightly higher specific gravity of the mica caused it to report

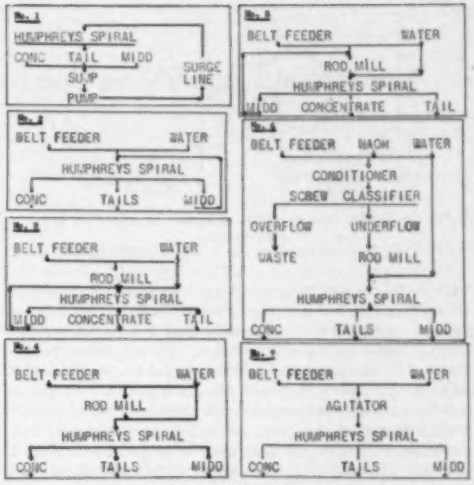


Fig. 2—Flowsheets for Humphreys spiral tests.

in the bottom of the tubes. Flowsheets for Humphreys spiral tests are given in Fig. 2.

In general, the extent of feed preparation necessary for efficient spiral separation depends on the natural degree of delamination of the mica flakes. Most samples tested responded poorly to direct spiral treatment, but one exception to this rule makes further investigation necessary on the subject, Table I, Test 1, with a concentrate grade of only 45.8 pct mica, as compared with Tests 12 and 13 with concentrate grade up to 91 pct mica. In this case, since the mica is already in thin flakes, it is feasible to concentrate it directly. Since the spiral concentration of mica is based solely on shape difference, most ores require a pass through some form of reduction equipment to delaminate the mica and accentuate its shape difference. The feed preparation should be

Table I. Test Data

Test No.	Lab. Sample No.[a]	Flowsheet[b]	Feed Preparation	New Feed, Lb per Hr	Solids in Rod Mill, Pct	Circulating Load, Pct	SPIRAL FEED: Lbs per Hr	SPIRAL FEED: Solids, Pct	Recov. Mica, Pct	Mica in Sp. Conc., Pct	Mica in Feed, Pct
1	649	1	none				683	10	72.6	45.8	18.0
2	650	2	none				1000	10	85.1	74.5	33.3
3	670	4	rodmill	1000	35		1070	12	48.6	58.0	16.1
4	671	3	rodmill	890	34		890	12	63.6	58.9	16.5
5	663	4	rodmill	434	51		434	17.7	82.4	52.3	9.0
6	663, 715	5	rodmill	451	39	35	607	26.9	82.7	80.0	11.6
7	663, 715	5	rodmill	446	41	27	566	15.3	83.6	63.0	11.6
8	715, 748	5	rodmill	449	50	170	1212	50.0	80.8	76.3	17.7
9	703	6	rodmill	356	52		356	20.0	90.8	74.2	12.0
10	706	4	rodmill	313	51		313	13.8	74.0	93.8	42.1
11	706	5	rodmill	326	37	31	427	15.1	83.0	87.5	42.1
12	749	7	screened through 16m				986	26	68.5	86.0	17.5
13	749	7	screened through 16m				949	44.2	85.4	91.2	17.5

[a] Laboratory Sample Nos.
637, 670, 671 —⅛ in. pegmatite-type waste from all sections (wet and dry) of the Chas. Bradley Mining Co. mica recovery plant in Jackson County, N. C.
649 —⅛ in. alaskite-type waste from the wet section, (4 trommels and 3 sets of rolls), of the Fred Deneen Mica Co. plant in Yancey County, N. C.
650 10-mesh alaskite-type waste from the dry section (1 set of rolls and 1 trommel screen), of the Fred Deneen Mica Co. plant in Yancey County, N. C.
663, 715, 748 —⅛ in. alaskite-type waste from the entire plant of the Southern Mica Co., Yancey County, N. C.
703 Unstained granitic-type ore taken from below surface zone, Hart County, Ga.
706 —⅛ in. rougher mica (pegmatite type) concentrate from the wet section of the Chas. Bradley Mining Co. plant in Jackson County, N. C.
749 —⅛ in. granitic-type waste from the entire mica recovery plant of F. B. Hendricks in Cleveland County, N. C.

[b] See Fig. 2.

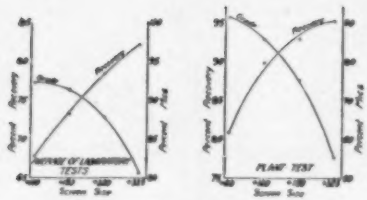


Fig. 3—Grades and recoveries obtained by screening spiral concentrates.

such as to reduce the size of gangue particle to about 16-mesh. There is some evidence that this mesh size can be extended to 10-mesh under special conditions. The size of mica flakes should not exceed about 1/3 in., since large flakes tend to catch on the edge of the ports. In practice any clean, coarse mica would normally be scalped off ahead of the spiral.

Desliming of the feed ahead of the spiral is not essential but may be desirable in order to facilitate the setting of ports and wash water. Most of the slime accompanies the mica and must be removed at some point in the process. If desliming follows the spiral, slime and water may be removed at the same time on a fine mesh dewatering screen.

Rod milling as a means of feed preparation was chosen as the most efficient means of grinding to 16-mesh without producing excessive slimes. The rod action appears to be very effective in delaminating the mica, as shown by the light weight, 19.6 lb per cu ft, of +100-mesh spiral concentrate, determined by Scott volumeter (concentrate from Test 9).

Following adequate feed preparation, the next important variable tested was feed density, which was found to be closely related to tonnage rate. Initial tests seemed to show that a low density feed, ranging from 10 to 20 pct solids gave the best results. Later laboratory tests showed that density as high as 50 pct solids was advantageous, see Tests 5, 6, 7 and 8 in Table I. The first plant test showed that feed density as high as 50 pct solids could be employed, providing the tonnage rate was sufficient to give a reasonable flow over the spiral, and the mesh size of the feed was sufficiently fine. Because of the large capacity of the standard Humphreys spiral, it has been impossible in the laboratory with present equipment to check the higher densities with their necessary high feed rates.

From discussion with engineers of the Humphreys Investment Co., it was learned that the capacity of a spiral can best be expressed in gallons per minute. This capacity will vary with the material being treated but will be fairly constant for any particular feed. Capacity figures vary from 10 to as high as 35 gpm. Because of the bulky nature of this mica feed, it was evident that the capacity would be in the lower part of this range. If 10 gpm may be considered a desirable feed rate to one spiral, the importance of using a feed density as high as possible is very apparent. At 50 pct solids this would give 1.85 tons per hr, while at 20 pct solids, feed would be only 0.58 tons per hr. At the time of this writing, feed rates have not been sufficiently steady at the first plant installation to make accurate testing possible, but from the results so far obtained, it appears that a rate of about 1½ tons per hr per spiral will be possible at pulp density approaching 50 pct solids.

Middlings Recirculation

The subject of recirculating of middlings has been carefully studied, and quite definite conclusions have been reached. With all types of mica ore tested (see Table I for description of samples) there has proved to be an advantage in returning the middlings either to the spiral or, preferably, to the grinding unit. A high proportion of the spiral middlings is composed of "booky" particles of mica, "platey" particles of gangue and, in some cases, particles of semi-kaolinized feldspar. An additional pass through the grinding circuit is necessary to further delaminate the mica particles and reduce the particle size of the other constituents. The benefits of returning the middlings may be seen from Tests 5, 6 and 10, 11.

Screening of Concentrates

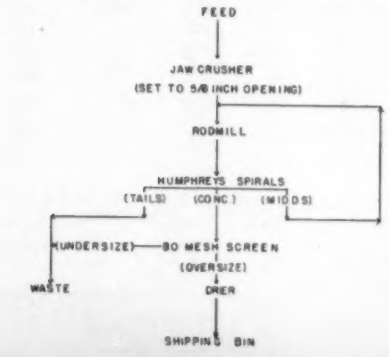
The mica concentrate from spiral operation invariably contains some fine sand and usually some clay slimes. Since this mica concentrate is very dilute (under 10 pct solids), some form of dewatering screen must be employed. The choice of mesh size for this screen is determined by the compromise between recovery and concentrate grade. Curves in Fig. 3 show the effect on recovery and grade of screening these mica concentrates at various mesh sizes.

Summary

As pointed out earlier, the initial work was focused on finding a method of treating the rejects from conventional washing plants. It soon became apparent that for simplicity and low costs, the ideal installation would eliminate the rolls and screens by taking the feed directly to the rodmill, or in the case of pegmatites, follow the jaw crusher with a rodmill as shown in Fig. 4. Tests on crude ore treated in this way showed good recovery of acceptably clean mica.

In conclusion, it is believed that the process described has been sufficiently proved to justify its installation in the field. As an addition to existing plants it will recover about three quarters of the mica now being wasted, see Fig. 1. If installed as in Fig. 4, the process will successfully replace existing methods, giving a simpler, cheaper and more efficient means of recovery.

Fig. 4—Simplified flowsheet for a small mica recovery plant.



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A New Surface Measurement Tool for Mineral Engineers

by F. W. Bloecher, Jr.

DETERMINATION of the surface area of finely divided minerals is of interest to mineral engineers engaged in flotation, comminution, and dewatering studies. In the industrial minerals field, many materials are classified as to fineness according to their specific surface (surface area per unit weight). A method of surface area determination has been tested and used in the Richards Mineral Engineering Laboratories at M.I.T. It involves low temperature krypton adsorption measurements using the technique described by Beebe.[1]

During the past fifteen years a number of new surface measurement techniques have been developed. Those that are broadly applicable to materials dealt with by mineral engineers include the permeability method[2-5] in which a gas or liquid is passed through the sample, and the low temperature gas adsorption methods first described by Brunauer, Emmett, and Teller.[6-10] Efforts to make surface determinations from size distribution curves have long been attempted[9,11,12] but because of the uncertainty of sub-sieve size distribution and inaccuracies in screen analyses, the method is at best only a rough approximation.

The most widely used gas adsorption method utilizes nitrogen gas.[6-8] Nitrogen adsorption, however, is not particularly suited as a tool for most mineral engineers because the method is not applicable for surface determinations on coarse materials, that is, materials coarser than approximately 150 to 200-mesh. Other gases have been used in mineral engineering studies, particularly ethane.[6, 10] The use of ethane gas, however, involves a technique that is too sensitive for most mineral engineering work.

The permeability method of surface area determination has the disadvantage of not being sensitive enough to detect the surface represented by minute cracks, the so-called internal surface of a particle.

The low temperature krypton adsorption method described by Beebe[1] utilizes apparatus and techniques that are essentially the same as those used in the ethane adsorption method.[6, 10] The surface area is calculated by the B.E.T. method.[6-10] The krypton adsorption technique is particularly suited to the needs of the mineral engineer because:

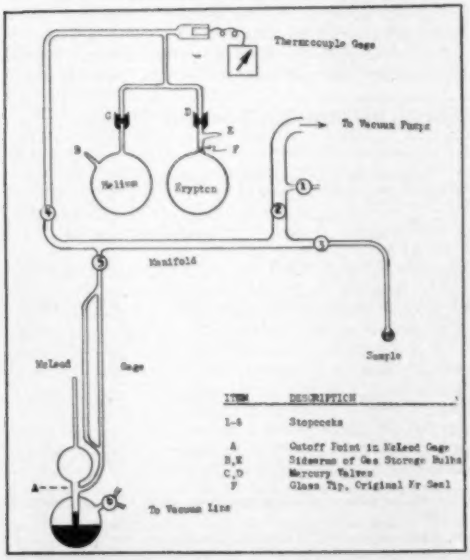
(1) The method can be used for making surface area measurements of rather coarse materials (35 to 48-mesh), as well as extremely fine materials, such as clays or colloidal slimes with specific surface that might be as high as 20 to 30 m^2 per g.

(2) The use of krypton gas offers the advantage of low saturation pressures (2-3 mm) at the conveniently obtained liquid nitrogen temperature. Therefore the krypton vacuum system need not be designed to measure or withstand internal gas pressures above one atmosphere as is the case with the nitrogen gas method.

(3) The krypton technique offers the safety feature of employing liquid nitrogen, as contrasted to the ethane technique in which liquid oxygen is required. Furthermore, ethane vapor pressure at liquid oxygen temperatures is low (about 10 microns), and a high sensitivity McLeod gage is required for pressure readings.

(4) The measurement technique with krypton gas is much simpler and faster, and the apparatus is less elaborate than that required with either nitrogen or ethane gases.

Fig. 1—Krypton surface measurement apparatus.



Apparatus

A krypton surface measurement apparatus like that used at M.I.T. is shown in Fig. 1. It is constructed of pyrex glass. A vacuum is maintained by two mercury diffusion pumps in series backed by a mechanical vacuum pump.[13] The vacuum thermocouple gage is used for checking the vacuum within the system. This type of gage is quite sensitive down to a pressure of 1 micron of mercury,[14] but may be used to indicate pressures much lower than that

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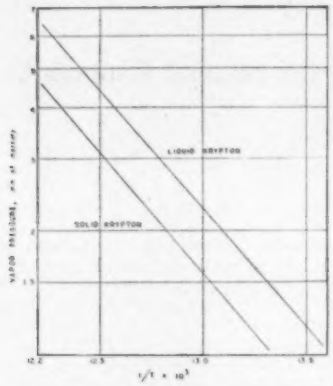

Fig. 2—Variation of krypton vapor pressure with temperature: log p vs. 1/T.

figure. If the needle on the thermocouple gage does not waver when the system is isolated from the vacuum pumps by turning stopcock 2, then the vacuum is near the required level. Such a technique, therefore, eliminates the use of more sensitive and delicate gages.

The stopcocks are a high-vacuum type with hollow plugs that are drawn into their shells by the vacuum within the system. Although high-vacuum stopcocks are more expensive than the standard type, their use will reduce, if not completely eliminate, time spent in searching for slight leaks. Stopcock 6 is a standard three-way type used for raising or lowering the mercury in the McLeod gage. One arm is connected to a water aspirator or laboratory vacuum line. Stopcock 1 is used for admitting air to the system and for evacuating the gas storage bulbs (via rubber tubing to sidearms *B* or *E*) prior to filling with helium or admitting krypton.

The mercury valves, *C* and *D*, utilize the surface tension of mercury to prevent the passage of gas out of the storage bulbs. They consist of two fritted glass faces, one attached to the storage bulb and the other to the system, both immersed in a pool of mercury. The surface tension of the mercury prevents it from passing through the frit and also prevents the gas from escaping. When the two fritted faces are placed in contact, the mercury between them is displaced, and the gas passes slowly through the frit into the system. Small quantities of gas thus can be admitted to the system at a very slow rate.[18]

Krypton gas can be purchased from the Air Reduction Co. in a sealed pyrex storage bulb. The lower half of the mercury valve and a side arm *E*, must be attached to the neck of the bulb. A small iron nut is placed in the side arm. The arm is connected to the system at stopcock 1 by pressure tubing, and all the air is pumped out of the neck of the krypton bulb. The side arm is then sealed off by heating with a torch. The small tip *F* sealing the krypton in the bulb is broken by manipulating the iron nut with a hand magnet.

The helium is ordinary tank grade (99.8 pct pure) and is admitted to the bulb through pressure tubing connected to the tank. A glass tee placed in the tube leading from the tank will provide means for a hose connection to the system at stopcock 1, thereby enabling the air to be evacuated from the bulb and tube prior to admitting helium. The helium is passed through a trap immersed in liquid nitrogen in order to remove any water vapor or other condensable gases which it might contain. Helium pressure in the storage bulb must not be allowed to exceed one atmosphere, so that the side arm *B* can be sealed off by heating with a torch.

The low-sensitivity McLeod gage has been calibrated linearly[16] so that pressures from 0.1 mm to 2.7 mm can be measured with it to three significant figures. The volume of the bulb of the gage (above cut-off *A*) was determined quite accurately by filling with mercury and weighing before it was placed in the system. With its volume known (41.78 cc), the gage can be used as a buret for determining volumes of other parts of the system and for measuring quantities of krypton gas used in the adsorption measurements. For this purpose the fundamental gas relationship, $P_1V_1 = P_2V_2$ is used.[9] The manifold volume of the system illustrated (that volume bounded by stopcocks 2, 3, and 4, and cut-off *A*) was 33.61 cc. The reproducibility of a series of such volume determinations was found to be of

Table I. Krypton Adsorption Data on 1.0302 G of Specular Hematite, Cone 5 Infrasizer Fraction

	1	2		3	4	5	6	7	8	9	10
Step	Description	Room Temp. °C	Room Temp. °K	Krypton Pressure, P_1[a]	Amount of Kr, cc-mm, P_1V_1[b]	Adsorption Pressure, p	Amount Kr in V_2 cc-mm, pV_2[b]	Kr Adsorbed, cc-mm, $P_1V_1 - pV_2$	Kr Adsorbed, cc at STP, V	$\frac{p}{V(P_o-p)}$	$\frac{p}{P_o}$
1	Kr in	24.4	297.5	2.14	161.33	1.37	106.52	54.81	0.06620	11.2[e]	0.425[e]
2	Kr out				—103.28[c]						
3		24.4	297.5		58.05	0.430	33.66	25.39	0.03066	4.89	0.130
4	Kr out				—14.12[b]						
5		24.5	297.6		43.93	0.307	23.87	20.06	0.02422	4.35	0.0953
6	Kr out				—23.14[c]						
7	Net Kr				20.79						
8	Kr in	24.6		1.38	104.04						
9	Total Kr	24.6	297.7		124.83	1.04	80.86	43.97	0.05307	8.99	0.323
10	Kr out				—34.95[d]						
11		24.6	297.7		89.88	0.719	58.75	34.13	0.04119	6.95	0.223
12	Kr out				—34.10[d]						
13		24.5	297.6		65.78	0.494	38.41	27.37	0.03304	5.48	0.153

[a] Pressures are in mm of mercury.
[b] V_1 = 75.39 cc (except in steps 4, 10 and 12 where V_1 = 33.61 cc) V_2 = 77.75 cc.
[c] Kr pumped out of volume V_1; i.e., in step 2, 75.39 x 1.37, or 103.28 cc-mm of Kr removed.
[d] Kr pumped out of manifold volume only; i.e., in step 4, 33.61 x 0.420, or 14.12 cc-mm of Kr removed, (33.61 cc = manifold volume).
[e] P_o = saturation pressure of super-cooled liquid Kr, 3.22 mm. Observed pressure of solid Kr was 2.25 mm (see fig. 3).

the order of ±1 pct. Helium gas was always used in making volume measurements.

Krypton Adsorption Data

The technique of obtaining krypton adsorption data is similar to that used by others with the ethane apparatus.[5, 10] A representative sample is weighed out in a suitable pyrex bulb and sealed to the system with a glass blowing torch. A total sample surface of at least 1000 cm^2 is required to insure good experimental reproducibility. If the sample contains adsorbed water, as is the case with certain clays, it should be heated at about 100°C until it reaches a constant dry weight before it is sealed to the system.

After the sample is attached to the system, the air is pumped out of the sample chamber. The initial evacuation of air should be started with extreme caution by carefully opening stopcock 3. If this stopcock is opened too suddenly, the sample will be sucked into the system. The sample is heated to about 100° to 200°C with a small electric furnace and evacuated for several hours to remove all adsorbed gases. The thermocouple gage can be used to measure the completeness of this "outgassing" of the sample surface. Three to four hours evacuation will usually suffice.

After evacuation the heat source is removed and the sample is allowed to reach room temperature. The combined manifold, gage and dead space volume is determined using the McLeod gage as a buret (the dead space is the volume surrounding the sample beyond stopcock 3). The gage and manifold volumes, of course, will be constant for a given vacuum system, but the dead space volume will change every time a new sample is sealed to the system.

Before starting krypton adsorption measurements, the sample must be cooled to liquid nitrogen temperature. To speed up this cooling, helium gas (approximately 1 cm pressure) is admitted to the dead space to act as a heat transfer medium. Ten to fifteen minutes were found to be sufficient to insure that the sample temperature had reached the desired level. With the sample bulb remaining in liquid nitrogen, the helium is then completely pumped out. No helium is adsorbed by the sample at liquid nitrogen temperature.

Some krypton is admitted to the system through the mercury valve D, stopcock 3 being closed. It occupies a volume V_1, consisting of the gage plus manifold volumes, and exerts a pressure P_1, as noted in line 1 of Table I, a collection of adsorption data on specular hematite. There are a total of P_1V_1 cc-mm of krypton occupying the volume V_1, as recorded in col. 4 of Table I. By opening stopcock 3 the krypton is allowed to expand into the dead space, the sample tube still being immersed in liquid nitrogen. After allowing about 5 to 10 minutes for adsorption equilibrium to set in, the adsorption pressure p is measured with the McLeod gage and recorded (col. 5). The total amount of gaseous krypton remaining in volume V_2 (gage plus manifold plus dead space) is then pV_2, cc-mm, recorded in col. 6. The difference between the values of col. 4 and 6, which is recorded in col. 7, is the cc-mm of krypton adsorbed by the sample at the adsorption pressure p. The factor, $\frac{273.1}{°K} \times \frac{1}{760}$, is used to convert the cc-mm of col. 7 to the cc at standard temperature and pressure (STP) recorded in col. 8. The pressure, P_o, used in

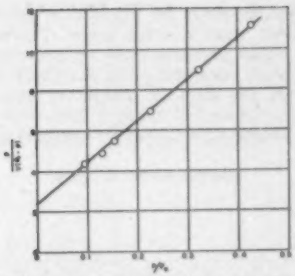

Fig. 3—B.E.T. plot of krypton adsorption on specular hematite.
Slope = 26.5
Intercept = 2.37
Area = 3230 cm^2 per g

calculating the values in col. 9 and 10 is the saturation pressure of liquid krypton at the temperature of absorption and is not measured until after a series of adsorption measurements have been completed, i.e., until after the data of line 13 has been collected.

After measuring the adsorption pressure p of line 1, stopcock 3 is closed, and the krypton remaining in the volume V_1 (gage plus manifold) is completely pumped out. That is, pV_1 cc-mm of krypton are removed from the system, as shown in line 2. The remaining krypton is allowed to redistribute itself throughout the volume V_2 and the adsorption data of line 3 is collected as was that of line 1.

Stopcock 3 is closed again to allow for the krypton removal step of line 4. However, in this gas removal step, the krypton is pumped out of only the manifold volume, the mercury being allowed to remain above the cut-off in the McLeod gage, thereby retaining the krypton in the gage volume as well as that in the dead space.

Then another set of adsorption readings is obtained and recorded as per line 5.

The rest of the data in Table I is obtained in a similar manner, by addition of more krypton (line 8) followed by alternate expansion and evacuation.

After the adsorption data is collected, krypton gas is admitted to the system until it is visibly condensed on the walls of the sample tube. The vapor pressure of this condensed krypton is measured allowing sufficient time for pressure equilibrium to set in. According to Beebe,[1] it is the vapor pressure of supercooled liquid krypton which should be used in the

Table II. Some Representative Surface Area Measurements

Material	Weight, g	Description	Surface Area, cm^2 per gram[a]
Monazite conc., a	1.314	48/100-mesh	2,140
Monazite conc., b	2.423	48/100-mesh	1,980
Monazite	0.1582	1-hr grind	13,550
Monazite	0.1278	2-hr grind	23,700
Monazite	0.1301	4-hr grind	42,400
Monazite	0.1060	8-hr grind	69,000
Monazite	0.1044	18-hr grind	135,000
Smithsonite	1.671	100/270-mesh	5,440
Smithsonite	1.671	100/270-mesh	5,300
Smithsonite	1.671	100/270-mesh	5,300
Smithsonite	1.671	100/270-mesh	5,350
Quartz	1.257	—200-mesh, deslimed at about 400-mesh	897
Sphalerite	10.14	65/100-mesh	226
Sphalerite	10.14	65/100-mesh	225
Ilmenite-Leucoxene conc.	2.294	48/100-mesh	42,100
Ilmenite-Leucoxene conc.	0.2621	48/100-mesh heated in 1:1HCL for 1 hr, water washed and dried.	57,700
Specular Hematite	1.630	Cone 4 infrasizer fraction	3,230

[a] The area per krypton molecule was taken as 19.5 sq A.

B.E.T. equation, since the krypton is adsorbed as a liquid rather than a solid layer. Therefore, the measured vapor pressure must be converted to the liquid krypton pressure using the chart of Fig. 2, drawn from the data of Meihuizen and coworkers.[17, 18] The adjusted P_o value is then used in calculating the figures in cols. 9 and 10 of Table I.

Interpretation of Data

The B.E.T. equation is usually written in the form:[6, 7]

$$\frac{p}{V(P_o - p)} = \frac{1}{V_m C} + \frac{(C\text{-}1)\,p}{V_m C\,P_o} \qquad [1]$$

where V is the volume of krypton (cc at STP) adsorbed at a pressure p, P_o is the vapor pressure of super-cooled liquid krypton at the temperature of adsorption, V_m is the amount of gas (cc at STP) required to form a monolayer on the sample, and C is a constant related to the heat of adsorption of krypton.

By making a plot of the experimental values of $\frac{p}{V(P_o - p)}$ as the ordinate, against p/P_o as the abscissa, a straight line results over a p/P_o range of about 0.05 to 0.35. The slope of that line will be equal to $\frac{(C\text{-}1)}{V_m C}$, and the intercept equal to $\frac{1}{V_m C}$. Solving the two equations simultaneously results in the simple equation for V_m of:

$$V_m = \frac{1}{\text{slope} + \text{intercept}} \qquad [2]$$

Fig. 3 is a B.E.T. plot of the data in cols. 9 and 10 of Table I. The slope of the line is 20.5, and its intercept 2.37. Substitution of those figures in eq 2 gives 0.04373 cc of krypton as the amount required to form a monolayer of krypton on the surface of the hematite sample.

The specific surface of the sample is calculated using the equation:

$$S = \frac{V_m \times 6.023 \times 10^{23} \times 10^{-16} \times d}{22400 \times \text{wt of sample}} \qquad [3]$$

where 6.023×10^{23} is the number of molecules in 22,400 cc of krypton gas at STP (Avogadro's number), d is the cross-sectional area of a krypton molecule in square angstroms, and 10^{-16} is the factor converting square angstroms to cm^2. According to Beebe's data,[1] the area per krypton molecule is approximately 19.5 sq A. That figure has been checked closely by making a surface area determination on a sample of zinc oxide previously measured by the nitrogen adsorption technique. Beebe's figure of 19.5 sq A has been used for the work reported herein.

Inserting the weight of the hematite sample and the calculated value of V_m in eq 3 gives a specific surface of 2230 cm^2 per g for that material.

Typical Surface Measurements

Table II contains a representative list of materials on which surface area determinations have been made with the krypton apparatus. A glance at the table will show that the method is quite well suited for use in the mineral engineering laboratory. The coarsest material on the list, from a surface area standpoint, had a specific surface of only 225 cm^2 per g, while the finest material measured had an area of 135,000 cm^2, or 13.5 m^2, per g. The series of runs on the 1.671 g sample of smithsonite shows that the reproducibility of experimental data is well within + or − 2 pct. The two runs on the coarse sphalerite show that the experimental reproducibility is excellent even when dealing with rather coarse materials.

Acknowledgment

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Application of Cyclone Thickeners To Preparation Plant Water Circuits

by G. H. Kennedy and H. E. Criner

CYCLONE thickeners are "hydraulic centrifuges" designed either to produce a maximum practical separation between the water and solids of a feed slurry or to classify the slurry solids according to particle settling rate. The possibilities of utilizing the thickener to "close" plant water circuits have been investigated.

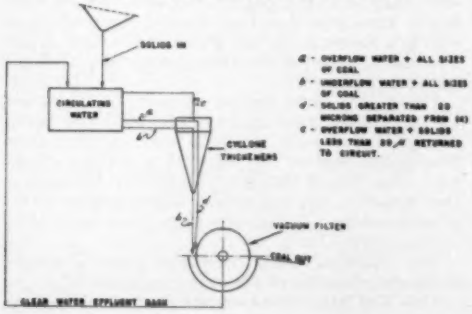
Since the thickener becomes an integral part of the plant water circuit when installed, the basic means by which the solids content of all circulating water systems must be controlled is reviewed.

All coal not separated from the circulating water by means of dewatering screens or settling devices must be bled from the plant at the same rate as they are being fed or created by degradation. If this were not true, the circulated volume would finally be completely filled with solids.

If the rate of bleed is low, the solids concentration in the bleed and circulating water generally will be high. To keep a reasonable circulating water concentration, large amounts of fresh water must be supplied and the bleed rate must be high. These conditions are usually undesirable because of the difficulty in securing fresh water, the loss of large quantities of coal, and the pollution of nearby streams.

Thickeners are the primary stage in the separation of the water contained in the bleed from the coal. Since a thickener, when operating with reasonable efficiency, usually cannot be expected to deliver an underflow containing less than 50 pct water, secondary dewatering devices such as a slurry screen or vacuum filter must be used to separate additional water from the bleed. The surface area required by the filter or screen will be nearly proportional to the total water which must be handled, and therefore the thickener becomes necessary to hold down the original cost. The water separated from the coal is then returned to the plant circuit to maintain the water balance and thus becomes the source of fresh water.

Fig. 1—Elements of closed-circuit water system.


SOLIDS IN
CIRCULATING WATER
CYCLONE THICKENERS
VACUUM FILTER
COAL OUT
CLEAR WATER EFFLUENT DAM
a - OVERFLOW WATER + ALL SIZES OF COAL
b - UNDERFLOW WATER + ALL SIZES OF COAL
d - SOLIDS GREATER THAN 50 MICRONS SEPARATED FROM (c)
c - OVERFLOW WATER + SOLIDS LESS THAN 50 μ RETURNED TO CIRCUIT.

It is not necessary to return clear water to the plant to maintain control of the circulating water concentration. It is only necessary that the bleed dewatering devices remove coal in all sizes at a rate equal to that at which these sizes are entering the circulating water. However, practically speaking, in order to maintain low solids concentration, it is usually necessary that the thickener have good recovery characteristics on bituminous coals in those sizes down to 400-mesh.

The thickener can remove the smaller sizes from the closed-circuit system at the "solids in" rate and prevent their "build up" in the circulating water as shown in Fig. 1. With the portion of the feed water going directly to the underflow, comes the very fine solids not concentrated by the cyclone. All the larger solids are removed from the feed and delivered in a concentrated condition at the underflow.

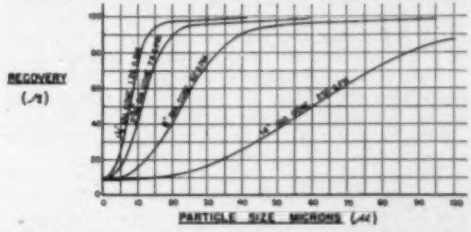


Fig. 2—Relative recovery efficiencies of various sizes of cyclone thickeners.

To apply thickeners with some reasonable assurance that operating requirements will be met, it is necessary to know the rate at which solids of all sizes must be removed from the circulating water and to know the thickener characteristics.

The required removal rate of fine solids not recovered by other dewatering devices may be estimated from the size distribution and tonnage of the plant feed with a reasonable allowance for degradation in the circulating water system. A more reliable estimate may be secured by sampling the plant bleed water during a constant load period.

G. H. KENNEDY, Preparation Engineer, is with the Rochester and Pittsburgh Coal Co., Indiana, Pa. H. E. CRINER, Development Engineer, is associated with Heyl and Patterson, Inc., Pittsburgh, Pa.

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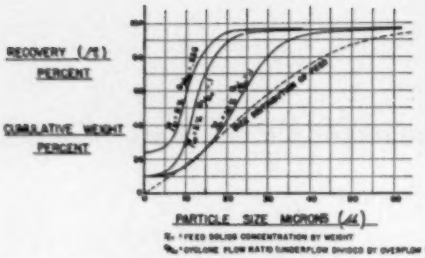

Fig. 3—Influence of feed concentration and flow ratio on recovery of 2 13/16-in. diam cyclone thickener.

The thickener characteristics will depend upon the design of the unit and the operating conditions. Fig. 2 shows the influence of one design factor, i.e., size. Fig. 3 shows the effect of feed concentration and flow ratio. It is to be noted that increasing cyclone size, feed concentration, and decreasing flow ratio will all serve to lower thickener efficiency.

If the above factors are known, a good estimate of the resultant circulating water and filter feed concentration and size distribution may be calculated.

A cyclone thickener, vacuum filter combination was installed in September 1949, at the Kent Plant of the Rochester and Pittsburgh Coal Co. This paper contains operating data obtained at this installation.

The preparation plant is a combination of sand flotation system washing 5x¼ in. coal and concentrating tables washing ¼ x 0 coal at a rate of 140 tph. The washed ¼ x 0 is collected in two settling tanks having a total volume of 65,000 gal connected about 2 ft below the water level. The water circuit of the coarse and fine coal plants are connected.

The effluent from the Centrifugal and Mechanical Industries centrifuges is returned to the settling tanks. The plant operates three shifts per day.

The ¼ x 0 coal is centrifugally and thermally dried; the ¾ x ¼ is thermally dried.

Initially the water clarification device was a solid bowl centrifuge. The feed consisted of a mixture of the effluent from the CMI's and circulating water. The feed rate was 400 gpm at 30 pct solids. The plant bleed was a portion of the effluent. The bleed rate was 100 gpm at 12 pct solids which were almost all smaller than 37 microns. The circulating water averaged about 25 pct; the cake averaged 30 pct moisture.

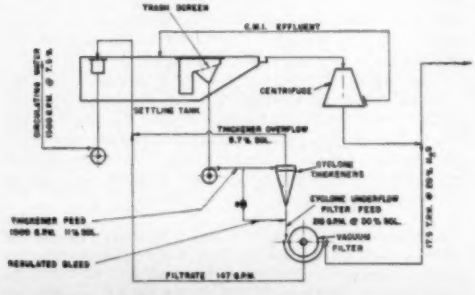

Fig. 4—Kent water circuit.

A detailed study, December 1947, showed that 17.5 tph of fines smaller than ½ mm must be removed in order to maintain an equilibrium of 7.5 pct solids in the circulating water and operate in closed circuit. It was decided that the plant should be so designed that it would be possible to dispose of an effluent containing no appreciable amount of coal.

Based on this study it was decided to install Heyl and Patterson cyclone thickeners and an Oliver disc type vacuum filter.

Fig. 4 is a flow diagram of the present circulating system. The cyclone feed passes through a ⅛-in. round hole, self-cleaning trash screen, which removes wood chips, waste and other foreign matter which would plug the cyclones. The feed pump supplies 1000 gpm at 40 psi to 4 banks of 3-in. diam cyclones, each containing 22 cyclones. The feed line to each

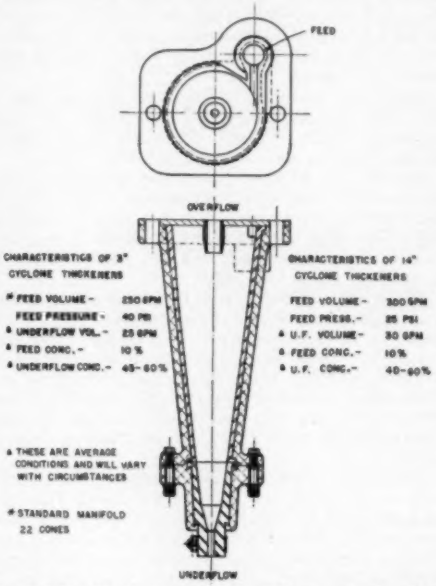
FEED
OVERFLOW
CHARACTERISTICS OF 3" CYCLONE THICKENERS
* FEED VOLUME - 250 GPM
FEED PRESSURE - 40 PSI
* UNDERFLOW VOL. - 25 GPM
▲ FEED CONC. - 10 %
* UNDERFLOW CONC. - 45-60 %
CHARACTERISTICS OF 14" CYCLONE THICKENERS
FEED VOLUME - 1000 GPM
FEED PRESS. - 25 PSI
▲ U.F. VOLUME - 30 GPM
▲ FEED CONC. - 10 %
* U.F. CONC. - 40-60 %
▲ THESE ARE AVERAGE CONDITIONS AND WILL VARY WITH CIRCUMSTANCES
* STANDARD MANIFOLD 22 CONES
UNDERFLOW

Fig. 5—Rubber-lined cyclone thickener, 3-in. diam aluminum body.

bank is valved so that any number of banks may be used. The underflow is pumped to a 8 ft 6 in. x 10 disk vacuum filter. The overflow from the thickeners joins the filtrate and is returned to the settling tanks at the point of the circulating water take off. The filter cake is discharged by way of a belt conveyor to the same conveyor that handles the centrifuged coal to a 25-ton surge bin ahead of two flash driers. The coal is conveyed from the bins to the flash driers by screw conveyors.

Fig. 5 is a cross section through the Heyl and Patterson 3-in. diam cyclone thickener. The casing is made in two parts, the main body and the underflow orifice holder. The material is die cast aluminum. The rubber lining is mounted in the case so that it can be replaced easily. The underflow orifice piece is rubber, and adjustment for wear is made by a simple hose clamp.

The following plant conditions have prevented complete operation of the cyclones:

(1) The bins ahead of the flash drier have a

tendency to plug when fed with much more than 12 tph of filter cake, so that it has never been possible to produce as much filter cake as required to bring the circulating water down to the concentration for which the system is designed.

(2) The overflow from the filter is returned to the sump ahead of the trash screen, resulting in recirculation of thickened material and excessive wear of the cyclones.

(3) The rate of fresh water input in the form of gland-sealing water has always been in excess of the water evaporated and lost on the coal surface so that there has always been a plant bleed.

For a special 5-hr test period the water circuit was modified to permit closed-circuit operation. During this time the following conditions were found to exist: circulating water concentration, 12.7 pct; cyclone thickener feed concentration, 18.6 pct; cyclone thickener underflow concentration, 55.6 pct; cyclone thickener overflow concentration, 11.5 pct; vacuum filter feed concentration, 37.2 pct; vacuum filter cake moisture (wet basis), 21.1 pct; and vacuum filter cake rate, 26 tph.

Fig. 6 shows the particle size distribution of the cyclone thickener products.

The increased filter cake rate can be attributed to an increased plant load of ¼ x 0 coal and a corresponding increase in —½ mm solids in the circulating water. When recalculated for this new condition, the equilibrium circulating water concentration would be 13.5 pct. Such close correspondence between calculated and actual results is possible when an understanding of the cyclone thickener characteristics is coupled with a detailed knowledge of the plant circuit and a knowledge of the rate at which solids must be removed from the system.

However, without the thickeners operating, the vacuum filter alone permitted the circulating water concentration to increase from about 13 pct to 25 pct within a three-shift period. Furthermore, when neither the vacuum filter nor the cyclone thickeners were in operation, the concentration of the water rapidly increased to 42 pct and necessitated a plant shutdown. The beneficial effect of the cyclone thickeners on the system is conclusive and the following are several specific improvements: (1) Positive control of the circulating water concentration has been made possible, (2) lower circulating water concentration has resulted in improved table performance,

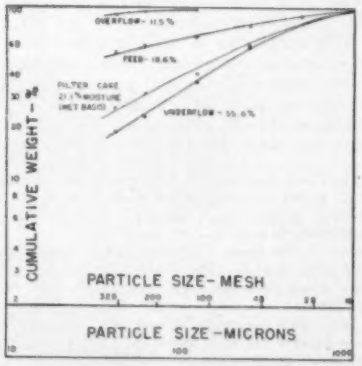

Fig. 6—Size distribution curves for 3-in. diam cyclone thickener products. Rochester and Pittsburgh Coal Co., Kent No. 1 and No. 2.

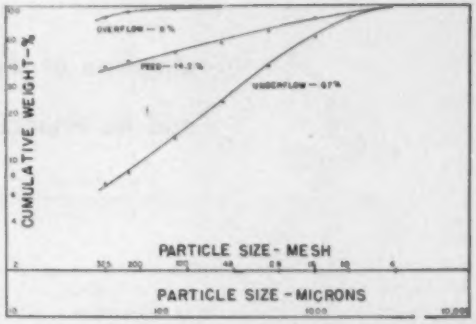

Fig. 7—Size distribution curves for 14-in. diam cyclone thickener products. Carpentertown Coal and Coke Co.

(3) fine coal lost to the silt ponds has been reduced, (4) abrasion rate in pumps and piping has been reduced, (5) pipe line plugging has been eliminated substantially, and (6) storage of concentrated slurry is at a minimum.

In addition to these operational improvements it is important to note the small space requirement for a cyclone thickener installation. Also, since it is within the plant structure, it is often possible to install it adjacent to the vacuum filter, permitting easy control of these interdependent operations.

Maintenance for the 3-in. cyclone thickeners consists chiefly of replacement of rubber parts subject to the abrasive action of the coal slurry. Field experience indicates that the rubber lining of the cone body will be serviceable for 1000 to 1500 hr depending upon the severity of the application. The rubber underflow orifice has a corresponding life of about 300 to 400 hr. Both parts are designed for rapid removal and replacement.

Another closed-circuit system using four 14-in. diam cyclone thickeners has been installed at Carpentertown Coal and Coke Co., Carpentertown, Pa. Through the courtesy of Mr. L. O. Lougee and Mr. C. Cornelius the following information is available.

Prior to the cyclone installation, fine solids were removed from the system on a 4x10-ft high speed dewatering screen having ¾-mm opening woven cloth and with a continuous plant bleed to a pair of settling ponds. The combined removal rate was estimated to be about 20 tph. Analysis of the solids to be removed revealed that only 5 pct passed 200-mesh and that 100 pct passed a 14-mesh sieve. This small percentage of very fine material made it possible to utilize the existing dewatering screen to further dewater the cyclone underflow. Thickening of the screen feed increased the screen retention to the point where closed-circuit operation was a reality. Test results showed the following operating performance: cyclone thickener feed, 14.2 pct; cyclone thickener underflow, 67 pct; cyclone thickener overflow, 8 pct; dewatering screen product moisture (wet basis), 25 pct; dewatering screen solids rate, 22 tph.

Fig. 7 shows the particle size distribution of the cyclone thickener products.

The two systems described serve to substantiate the claim that it is possible to calculate cyclone thickener capacity requirements, to estimate the performance with reasonable accuracy, and to close preparation plant water circuits in conjunction with dewatering screens or vacuum filters.

An Investigation of the Abrasiveness of Coal and Its Associated Impurities

by H. F. Yancey, M. R. Geer, and J. D. Price

COAL mine operators recognize coal as an abrasive material, because the wear of drilling, cutting, and conveying equipment is reflected as a cost item for replacement of parts. Similarly, industrial consumers of coal experience abrasive wear on all coal-handling equipment. Operators of pulverized fuel plants are doubtless most keenly aware of the abrasiveness of coal, because under the high contact pressures developed between coal and metal in pulverizers, abrasive wear is increased many fold. Moreover, experience in operating pulverized fuel plants has demonstrated that some coals are much more abrasive than others.

Hardgrove[1] stated that maintenance costs entailed by the wear of grinding elements is often a more important variable than the cost of the power required to pulverize different coals. Craig[2] also reports that one coal may cause pulverizer parts to wear several times faster than another. It is apparent, therefore, that those concerned with pulverizing coal could profitably employ a method for estimating the abrasiveness of different coals, just as they utilize standard tests for thermal value, grindability, and ash-fusion temperature to assist in selecting the most suitable and economical coal to use in a particular plant.

The objective of this investigation was to develop a test procedure that would be suitable for general use in estimating the abrasiveness of coals. However, few, if any, of the standard tests now used for evaluating the properties of coal are the product of a single investigation or the result of a single investigator's efforts. Rather, in each case, a testing procedure was devised by one investigator, used by others on a wider variety of coals, and finally refined completely as the result of the joint efforts of a number of interested people. Thus, the test procedure for estimating abrasiveness developed in the course of this work may not be refined sufficiently in its present form for general use, but it may serve as the starting point from which an acceptable test procedure can be developed. The method has been used thus far on only about a dozen coals, and there has been no opportunity to attempt a correlation between experimental results and actual plant experience. Only wider use of the procedure by other investigators and correlation with plant experience can determine to what extent the method will have to be modified to render it suitable for general application.

Test Method

Although the literature contains no record of an attempt to devise a method for estimating the abrasiveness of coal that could be used industrially, several investigators have tested properties of coal that are closely related to its abrasiveness. The abrasiveness of a material generally is considered to be related to its hardness, and hardness tests for coal have been employed by Heywood,[3] O'Neill,[4] and Mathes.[5] Also, the resistance of coal to abrasion, a property that presumably is related to the abrasiveness of coal, was measured by Heywood[3] and by Simek, Pulkrabek, and Coufalik.[6] All these investigators tested only individual pieces of coal. Since coal is a heterogeneous material having components of varying properties, tests of this type can yield results having little more than academic interest. Only a test method that utilizes a representative sample of coal can give results that are useful industrially.

The abrasion tests used for various other materials have been considered for adaptation to testing the abrasiveness of coal. The tests used for metals,[7-9] paving and flooring,[10] and rubber,[11] cannot be used because coal is not sufficiently abrasive.[4]

The present experimental work was begun before World War II and was conducted by three research fellows[12-14] working under a joint agreement between the University of Washington and the Bureau of Mines. After a great deal of preliminary work with a variety of apparatus and materials, a test procedure was developed which consisted of rotating a test disk 2½ in. diam in a steel mortar containing the coal sample. The shaft carrying the test disk at the lower end and a 100-lb load on the upper end was free to move vertically. The bed of coal in the mortar was kept fluid by low-pressure air admitted through a port near the bottom of the mortar. Measurable wear on an Armco iron disk could be obtained in this test procedure, but, despite extensive efforts to eliminate them, several major disadvantages remained in this test method. First, with most coals the amount of wear on the iron disk did not exceed a few milligrams. Second, a single type of disk was not applicable for all coals. A smooth iron disk gave satisfactory results with both bituminous and sub-bituminous coals, but hardly any wear with anthracite or coke. A disk having studs or projections gave more satisfactory abrasion losses with anthracite and coke and presented no operating difficulties with free-burning bituminous and sub-bituminous coals. It could not, however, be used with caking coals because these coals formed a

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This report gives the results of work done under a cooperative agreement between the U. S. Bureau of Mines and the School of Mineral Engineering, University of Washington.

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cake on the face of the disk, preventing abrasive wear.

During the war years this problem was recessed for more important work, and when it was resumed the disadvantages of the earlier test procedure seemed so great that the method was abandoned and a fresh approach made to the problem.

Equipment and Procedure

The present abrasion apparatus is illustrated in Figs. 1 and 2. Essentially, the test consists of rotating four removable wearing blades in a charge of coal for a fixed number of revolutions and then determining the loss in weight sustained by the blades during the test. This loss in weight, expressed in milligrams, is considered the abrasiveness of the coal.

The apparatus consists of a shaft-and-arm assembly to hold the wearing blades, a mortar that holds the charge of coal, and a drill press to provide rotation.

The blades are 2 in. long by 1½ in. wide, and are made from 16-gage Armco iron. The edge of the blade next to the wall of the mortar is curved backward with respect to the direction of rotation on a ⅝-in. radius. The purpose of this curvature is to crowd or *nip* the coal particles between the blades and the wall of the mortar. A clearance of 0.20 in. is maintained both between the blades and the bottom of the mortar and between the blades and the wall of the mortar.

The blades are carried on four radial arms spaced 90° apart. These arms, shaped to fit the curved blades and provide support for them, are held in a hub on the shaft by set screws so that the length of the arms and therefore the clearance between the blades and the wall of the mortar can be adjusted.

The shaft carrying the arms is turned to semi-hemispherical shape at the lower end to fit a thrust bearing. The upper end of the shaft fits into the spindle of the drill press, where it is held by set screws.

The mortar, turned from extra-strength pipe, is 8 in. ID by 9 in. deep. A simple brass thrust bearing screwed through the bottom of the mortar supports and centers the shaft. Adjustment of this bearing maintains a 0.20-in. clearance between the bottom edge of the wearing blades and the bottom of the mortar. Four ⅜-in. square bars fastened to the underside of the mortar bottom radially at 90° intervals fit into matching grooves in the table of the drill press to center the mortar under the spindle of the drill press and prevent the mortar from rotating. A tight-fitting cover on the mortar is provided with a clearance hole for the rotating shaft.

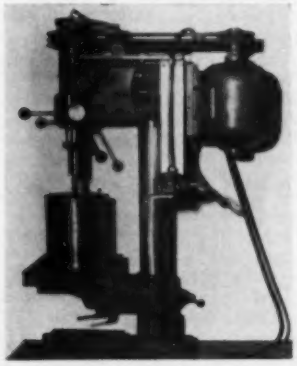

Fig. 1—Assembled abrasion apparatus.

Fig. 2—Mortar, shaft-and-arm assembly, and wearing blades.

The drill press is powered with a 3-hp motor and provided with sheaves to give a spindle speed of 1500 rpm. This amount of power is not required for ordinary testing but was necessary for tests in which the clearance between the wearing blades and the wall of the mortar was reduced substantially below 0.20 in. A Veeder-Root revolution counter to shut off the motor after a predetermined number of revolutions completes the abrasion apparatus.

In making a test, the four blades are weighed on an analytical balance to the nearest milligram. They are then fastened to the supporting arms, and the shaft assembly is placed in the mortar with the lower end of the shaft engaged in the thrust bearing. A 4-kg sample of air dried 4-mesh to 0 coal, prepared by stage-crushing with smooth rolls, is poured into the mortar.

After the rotating period, the wearing blades are removed, cleaned, cooled in a desiccator, and again weighed. With many coals, the blades can be cleaned adequately by merely rubbing with a dry cloth. Some coals, however, leave a residual film on the blades that is removed by rubbing with a damp cloth, then scouring with a fiber-brush wheel, and finally rubbing again with a damp cloth. The moisture from the damp cloth is evaporated quickly by the heat of the blades.

Coals Tested

Table I identifies the 14 coal samples tested by this test procedure. Most of these are from Washington, but a few representative coals from other states were included. These samples range in rank from anthracite to sub-bituminous B and cover a wide range in ash content. Float-and-sink fractions of a number of these samples were tested separately to illustrate the relative abrasiveness of the clean coal and its associated impurities.

Not all of the coal samples were tested after the operating variables of the test method were standardized, and consequently the data in some tables cannot be compared directly with similar data in other tables.

Influence of Variables

In developing a new test procedure, the exploration of the variables inherent in the procedure constitutes the major part of the investigation. Five principal variables in the present test procedure influence the magnitude of the abrasion loss ob-

Table I. Identification of Coal Samples Tested

State	County	Mine	Bed	Size[a]
Illinois	Christian	Langley No. 9	No. 6	Screenings
Pennsylvania	Indiana	Cush Creek	Upper Kittanning	Run-of-mine
Pennsylvania	Allegheny	Montour No. 10	Pittsburgh	3x0 in.
Pennsylvania	Schuylkill	Coaldale	Mammoth (?)	Pea
Utah	Carbon	Castle Gate	D	1¼x0 in.
Utah	Carbon	King	Hiawatha	Stoker
Washington	King	McKay	McKay	1x0 in.
Washington	King	Olson-Cumberland	Unnamed	Run-of-mine
Washington	Kittitas	Roslyn No. 3	Roslyn	Run-of-mine
Washington	Kittitas	Roslyn No. 3	Big	Run-of-mine
Washington	Pierce	Wilkeson-Wingate	No. 4 east dip	1¼x0 in.
Washington	Thurston	Tono	Tono	1⅛x⅛ in.
Washington	Whatcom	Bellingham No. 1	Bellingham	4x2 in.
Wyoming	Sweetwater	D. O. Clark	No. 9	Run-of-mine

[a] All coal samples were raw except those from the King, McKay, and Bellingham mines.

tained for a particular coal. They are: 1—The clearance between the wearing blades and the mortar; 2—the speed at which the blades are rotated; 3—the weight of the coal charge in the mortar; 4—the duration of the test period; and 5—the particle size of the coal tested.

If the wearing blades used in the test procedure are rotated in a charge of coal held in a container substantially larger than the mortar normally employed, little wear of the blades takes place. The crowding or squeezing action between the outside edges of the blades and the wall of the mortar is responsible for the abrasion sustained by the blades. Therefore, the clearance between the blades and the wall of the mortar is critical in determining the magnitude of the abrasion loss. If the clearance is too great, the abrasion loss is decreased; but if the clearance is too small the abrasion loss is likewise decreased. The optimum clearance to produce maximum abrasion doubtless is influenced by the maximum size of particles in the coal charge.

Table II shows the results of abrasion tests made on four coal samples with clearances between the blades and the wall of the mortar ranging from 0.10 to 0.30 in. These tests were made on coal having a top size of 4-mesh, or 0.185 in. It will be noted that, with all samples, the abrasion loss increased steadily with increase in clearance. Maximum abrasion loss with most coals probably would occur with a clearance of about 0.30 in. Nearly all of the results in this report were obtained in tests made with a side clearance of 0.20 in., and, consequently, do not represent the maximum abrasion loss that could be achieved.

Table II. Influence of Side Clearance on Abrasion Loss

Coal	Side Clearance, In.	Abrasion Loss	
		Mg.	Pct[a]
Olson-Cumberland	0.10	16	22
	0.15	62	83
	0.20	75	100
	0.25	82	109
	0.30	82	109
Roslyn No. 3	0.10	63	35
	0.15	151	85
	0.20	179	100
	0.25	199	111
	0.30	205	115
Montour No. 10	0.10	36	26
	0.15	96	69
	0.20	140	100
	0.25	141	100
	0.30	138	98
McKay	0.10	7	40
	0.15	14	74
	0.20	18	100
	0.25	22	122
	0.30	22	122

[a] Values expressed as percentage of abrasion loss obtained with clearance of 0.20 in.

The clearance between the bottom edges of the blades and the bottom of the mortar also influences the magnitude of the abrasion loss, as demonstrated in Table III, which shows results obtained in a series of tests on Olson-Cumberland coal where the bottom clearance was varied from 0.06 to 0.30 in.

Maximum abrasion loss was obtained with a bottom clearance of 0.20 in.

The side clearance required adjustment only when new wearing blades were substituted for badly worn ones. When used in testing ordinary raw coals, a set of wearing blades will last 25 to 30 tests before being worn down enough to increase the side clearance significantly. Adjustment of the bottom clearance was necessitated only to compensate for wear of the brass thrust bearing, and this adjustment was not required oftener than about every 50 tests. Substitution of a ball-bearing unit for the thrust bearing would eliminate any attention to bottom clearance.

With the equipment available, the rotation of the wearing blades could be varied through the range from 600 to 1500 rpm. The magnitude of the abrasion loss was almost directly proportional to the speed of rotation. For this reason, 1500 rpm was selected as the standard speed. No attempt to operate at higher speeds was made, despite the fact that speeds greater than 1500 rpm probably would give higher abrasion losses.

The mortar will hold a maximum charge of about 4 kg of a light coal, and this weight of charge was adopted as a standard. The data in Table IV show that, in the range from 1 to 4 kg, the abrasion loss is roughly proportional to the weight of coal.

Four kg is sufficient weight to constitute a representative sample of 4-mesh to 0 size coal. Use of a smaller quantity of coal of this size would not only

Table III. Influence of Bottom Clearance on Abrasion Loss

Bottom Clearance, In.	Abrasion Loss, Mg.
0.06	78
0.09	81
0.13	82
0.20	90
0.25	97
0.30	97

Table IV. Influence of Weight of Feed Charge on Abrasion Loss

Coal	Weight of Charge, Kg	Abrasion Loss	
		Mg.	Pct*
Roslyn No. 3	1	43	23
Roslyn No. 3	3	75	41
Roslyn No. 3	3	148	78
Roslyn No. 3	4	186	100
Langley No. 9	1	58	25
Langley No. 9	2	108	46
Langley No. 9	3	196	84
Langley No. 9	4	234	100
Bellingham	1	38	32
Bellingham	2	57	47
Bellingham	3	93	78
Bellingham	4	120	100

* Values expressed as percentage of abrasion loss obtained with 4-kg charge.

Table V. Influence of Duration of Test on Abrasion Loss

Coal	Abrasion Loss, Mg.		Increase, Pct*
	6000 Rev.	12,000 Rev.	
Roslyn Big Bed	50	84	68
McKay	17	26	53
Wilkeson-Wingate	23	33	43
King	51	89	78
Roslyn No. 3	78	136	74

* Percentage increase over abrasion values obtained at 6000 revolutions.

decrease the abrasion loss obtained, but also militate against the representativeness of the sample.

Test durations of 6000 and 12,000 revolutions have been used. As shown by the abrasion losses presented in Table V, doubling the length of the test does not double the abrasion loss obtained. With the five coal samples used, doubling the duration of the test increased the abrasion loss by an average of 63 pct.

Most of the abrasion on the wearing blades is caused by the coarser particles of coal. Since the coal is progressively pulverized during the course of the test, fewer coarse particles are present to cause wear during the later stages of a test of longer duration. Thus, the grindability of a coal influences its performance in the abrasion test. A coal that is easily crushed is soon reduced to the finer sizes and exhibits less abrasiveness, while a tougher coal is not pulverized as rapidly and exhibits abrasiveness over a longer period.

This relationship is illustrated in Table V in the comparison between the samples of Wilkeson-Wingate and King coals. The Wilkeson-Wingate coal is easily pulverized, and doubling the duration of the abrasion test increased the abrasion loss only 43 pct. The King coal is more difficult to pulverize, and it showed a 78-pct increase in abrasion loss when the test period was doubled.

The influence of the grindability of a coal on its abrasiveness, as shown by the present test method, appears to be one of the disadvantages of the test procedure. However, this disadvantage could not be overcome readily in any other testing method except one in which the size composition of the abrading coal was kept constant by the continuous addition of coarse particles and removal of fine material. This is a cumbersome test condition, difficult to establish.

The extent to which abrasion loss is influenced by the size composition of coal present in the charge is

Table VI. Influence of Particle Size Tested on Abrasion Loss*

Screen Size, Mesh	Screen Analyses of Coal Before and After Test, Pct							
	Size Tested 4-Mesh to 0		Size Tested 8-Mesh to 0		Size Tested 14-Mesh to 0		Size Tested 28-Mesh to 0	
	Before	After	Before	After	Before	After	Before	After
4 to 8	32.3	6.0						
8 to 14	31.1	21.2	45.9	24.1				
14 to 28	15.6	14.5	23.1	20.0	42.6	32.5		
28 to 48	9.3	8.3	13.8	13.3	25.4	23.0	44.3	33.5
48 to 100	5.9	5.4	8.7	8.1	16.1	15.4	28.1	28.1
100 to 200	3.1	6.2	4.5	5.1	8.5	7.0	14.7	16.5
Through 200	2.7	38.4	4.0	29.4	7.4	22.1	12.9	21.9
Abrasion Loss, Mg.	180		152		75		33	
Reduction in Average Particle Size During Test, Pct	69		38		19		17	

* Abrasion tests made on Roslyn No. 3 coal.

indicated in Table VI. A series of abrasion tests was made on Roslyn No. 3 coal with the standard side clearance of 0.20 in. The top size of material tested was 4, 8, 14, and 28-mesh. The abrasion losses obtained for these different sizes of coal range from 185 mg for the standard 4-mesh to 0 size to 33 mg for the material having a top size of 28-mesh. The particles coarser than about 14-mesh appear to be responsible for most of the abrasion.

The screen analyses of the coal before and after each test indicate the amount of grinding that took place during the test. The amount of grinding may be expressed as percentage reduction in average particle size, which varied from 69 with the 4-mesh coal to 17 with the 28-mesh coal. Thus, there is an approximate correlation between the amount of abrasion obtained and the amount of grinding that takes place.

The rate at which coarse particles are ground to finer size and the influence of this change in size composition on abrasion loss are demonstrated by the results presented in Table VII. This table summarizes the results obtained in a test made on Roslyn No. 3 coal in which all test conditions were standard, except that the apparatus was stopped at the end of each 1000 rev, and the size composition and the abrasion loss determined for each 1000 rev. The percentage of 4 to 8-mesh coal was reduced sharply during the first 1000 rev and then decreased at a much slower rate throughout the remainder of the test. A similar trend is evident with the 8 to

Table VII. Rate of Abrasion and Grinding During Test Period*

No. of Revolutions	Abrasion Loss, Mg.	Screen Analysis, Pct						
		4 to 8 Mesh	8 to 14 Mesh	14 to 28 Mesh	28 to 48 Mesh	48 to 100 Mesh	100 to 200 Mesh	Under 200 Mesh
0		30.2	33.3	16.0	9.3	5.4	3.6	2.2
1,000	15	20.9	28.5	15.9	9.8	7.8	5.6	11.4
2,000	12	18.0	27.5	14.8	10.6	7.4	6.1	15.6
3,000	13	13.7	27.7	16.6	9.9	7.5	5.8	18.8
4,000	12	12.3	27.5	16.1	9.8	6.3	6.5	21.5
5,000	13	12.8	27.3	15.9	9.2	5.9	5.3	22.6
6,000	13	13.6	26.6	15.3	8.3	5.5	5.1	25.6
7,000	12	10.5	28.3	15.3	7.9	5.6	5.2	29.2
8,000	10	10.9	25.6	14.9	8.3	4.9	5.2	30.2
9,000	9	10.2	25.1	15.3	7.9	5.2	5.2	31.1
10,000	9	10.2	24.9	14.6	7.8	5.0	5.4	32.1
11,000	8	9.1	26.0	15.1	7.4	4.6	4.8	33.0
12,000	10	8.8	24.9	14.9	7.6	4.7	4.7	34.8

* Test made on Roslyn No. 3 coal.

Table VIII. Screen Analyses of Soft and Hard Coals Before and After Abrasion Testing

Size, Mesh	Wilkeson-Wingate Float 1.40		Pennsylvania Anthracite	
	Before Test, Weight, Pct	After Test, Weight, Pct	Before Test, Weight, Pct	After Test, Weight, Pct
4 to 8	42.2	1.0	57.7	24.9
8 to 14	35.9	11.2	22.8	23.4
14 to 28	13.6	14.8	9.7	12.1
28 to 48	4.6	10.1	5.0	7.1
48 to 100	2.2	6.7	2.8	4.5
100 to 200	1.0	9.4	1.4	5.0
Under 200	0.5	46.8	0.6	23.0

Table IX. Relation Between Abrasion Loss and Moisture Content

Moisture Added, Pct	Total Moisture, Pct	Abrasion Loss, Mg.
0	3.1	163
.5	3.6	186
1.0	4.1	190
3.0	6.1	216

14-mesh material, although this size fraction was not depleted as much as the 4 to 8-mesh fraction. The proportions of the size groups between 14 and 200-mesh did not change significantly during the entire course of the test.

During the first 1000 rev the wearing blades lost 15 mg, and wear continued only at a moderately decreased rate during the next 6000 rev. The rate of abrasion fell somewhat after 7000 rev, but not sharply.

Roslyn No. 3 coal has a Ball-Mill grindability index of 35 pct, and thus is moderately difficult to grind. Naturally, with coals of high grindability, the coarser sizes are eliminated more rapidly than the rate indicated for Roslyn No. 3. Conversely, the coarse sizes disappear more slowly with coals of low grindability. The influence of grindability on the amount of coarse material remaining at the end of an abrasion test is shown in Table VIII, which gives the screen analyses of the float 1.40 sp gr fraction of Wilkeson-Wingate coal and a Pennsylvania anthracite before and after testing.

Table X. Results of Abrasion Tests Showing Precision of Test Procedure

Coal	Abrasion Loss, Mg.	
	New Blades	Worn Blades
King	163.6	174.7
King	158.4	176.6
King	159.2	177.3
King	161.3	182.2
Average	*160.4*	*177.7*
Wilkeson-Wingate float 1.60 sp gr	16.6	16.5
Wilkeson-Wingate float 1.60 sp gr	13.8	17.5
Wilkeson-Wingate float 1.60 sp gr	14.7	17.6
Wilkeson-Wingate float 1.60 sp gr	15.6	15.8
Average	*15.2*	*16.9*
Olson-Cumberland	87.6	
Olson-Cumberland	89.2	
Olson-Cumberland	87.7	
Olson-Cumberland	90.0	
Average	*89.2*	

With the Wilkeson-Wingate coal, which is easily pulverized, 12 pct of material coarser than 14-mesh remained at the completion of the test, while with the stronger Pennsylvania anthracite 48 pct of this size remained. The principal consideration in fixing the duration of the abrasion test at 12,000 rev was to insure that, with even the softest coal, some coarse material would be present in the charge at the end of the test.

Some of the discrepancies in the results of earlier abrasion tests directed attention toward the possible effect of the moisture content of the coal tested. Accordingly, a series of samples of King coal of different moisture contents was prepared by adding predetermined amounts of water to air-dried coal. After the moisture was added, the samples were placed in sealed containers, mixed well, and then allowed to stand overnight to insure a uniform distribution of moisture. Table IX shows the influence of these moisture additions on abrasion loss.

The air-dried coal, containing 3.1 pct moisture, gave an abrasion loss of 163 mg; addition of only 0.5 pct moisture increased this loss to 186 mg; and addition of 3 pct moisture increased the loss to 216 mg. Thus, the presence of even a small amount of excess moisture increases abrasiveness significantly.

With a change in moisture content of only 0.5 pct having such a marked effect on abrasion loss, it might be expected that the moisture change in air-dried coal due to changes in atmospheric humidity might affect abrasion results. But, samples of several coals have been tested repeatedly throughout several months of laboratory storage with consistent abrasion results.

The role played by moisture in the abrasiveness test is not clear. One explanation would be that the presence of even a small amount of water in the pores of the coal toughens it considerably. Under the conditions existing in the test procedure, corrosion resulting from the presence of the added water seems unlikely.

Precision of Test Method

The results of abrasion tests made in quadruplicate on three samples of coal are shown in Table X to demonstrate the degree of precision of the test procedure. The float 1.60-sp gr fraction of the Wilkeson-Wingate coal was selected to provide a material of low abrasiveness, the Olson-Cumberland coal is moderately abrasive, and the King coal is one of high abrasiveness. In the tests made with new wearing blades, the maximum difference between the four tests on Wilkeson-Wingate coal was 2.8 mg. The corresponding deviation for the samples of Olson-Cumberland and King coal was 2.4 and 4.2 mg, respectively. The 4.2 mg maximum deviation for the King coal represents 2.6 pct of the average abrasion value for this coal. This percentage deviation is probably no greater, for example, than that inherent in the standard friability test for coal. The 2.8 mg maximum deviation for the less abrasive Wilkeson-Wingate coal, however, amounts to 18.4 pct. Thus, with coals giving low abrasion losses, the precision of the method is lower than desirable.

The data in Table X also show a comparison between the abrasion results obtained with new and worn blades. The worn blades had been used long enough to sustain a total loss in weight of over 5 g. Ordinarily, blades are discarded after a total loss of 3 or 4 g, so the comparison shown is extreme. With both the King and Wilkeson-Wingate coals,

Table XI. Results of Abrasion Tests Made with Various Metals

Coal	Armco Iron	Mild Steel 1020	Type 301 Stainless Steel		Type 410 Stainless Steel
			Annealed	Full Hard	Annealed
Olson-Cumberland	89	79	71	78	78
Roslyn No. 3	163	163	137	146	141
King	163	149	131	144	140
Rockwell hardness B scale	42	62	88	118[a]	82

[a] A scale 75, roughly equivalent to B scale 118.

the badly worn blades gave abrasion results averaging about 11 pct higher than those obtained with new blades. Thus, if blades are discarded after being used for making enough tests to cause losses totaling not over 2 or 3 g, the influence of blade wear is probably within the accuracy of the testing procedure.

Although the accuracy of this test procedure, particularly with coals of low abrasiveness, is not as high as might be desired, it probably is good enough for the industrial application for which the method is intended. In other words, the industrial counterpart of the abrasion test, the wear of pulverizer grinding units, cannot be determined with great accuracy, and therefore a precise testing procedure probably is not required.

Comparison of Various Metals

Armco iron was originally adopted as the standard metal for the wearing blades for two reasons. First, it was thought that, being relatively soft, Armco iron would give higher abrasion losses. Second, Armco iron reputedly does not work harden and thus should offer constant resistance to abrasion. Comparison of the abrasion resistance of various metals is a complete metallurgical problem in itself and is outside the scope of the present investigation. However, SAE 1020 mild steel and several stainless steels in both the annealed and full-hard condition were tested to show how these materials compare with the Armco iron used as a standard. Table XI shows the results of tests made on three kinds of coal with wearing blades constructed of five different metals.

Surprisingly, SAE 1020 mild steel gave abrasion losses only moderately lower than those obtained with the standard Armco iron. The stainless steels likewise did not show the superior resistance to abrasion that had been expected. An interesting feature of these data is that the type 301 stainless steel was less resistant to abrasion in full-hard form than when annealed.

No significance is placed on the data shown in Table XI except that they appear to indicate that the use of Armco iron as a standard material for wearing blades does not result in abrasion values that are out of line with those obtained with the more common ferrous alloys.

Since work hardening of the wearing blades, if it occurred, would change their resistance to abrasion progressively, Rockwell hardness tests were made on several sets of blades before use, after the blades were about half worn out, and after the blades had been worn so badly that they were ready to be discarded. No increase in R hardness could be detected. Similar tests were made on the stainless steel blades. Although these blades were used for only a few tests and could not be tested after the extensive use given Armco iron blades, no work hardening was evident with these blades either.

Table XII. Results of Abrasion Tests on Various Kinds of Coal

Coal	Abrasion Loss, Mg.
Cush Creek	12
McKay	18
Wilkeson-Wingate	43
Tono	45
Olson-Cumberland	92
Bellingham	120
D. O. Clark	130
King	165
Montour No. 10	171
Roslyn No. 3	185
Castle Gate	212
Langley No. 9	242
Pennsylvania Anthracite	686
Sandstone	1210
Coke	2508

Abrasion Tests

Table XII summarizes the results of abrasion tests made on all the coal tested under standardized conditions. These standardized conditions were: Rotation of 1500 rpm; 12,000-rev duration of test; side and bottom clearance of 0.20 in.; Armco iron wearing blades; and 4-mesh to 0 coal.

Excluding the Pennsylvania anthracite, which gave an abrasion loss of 686 mg, these coals ranged in abrasiveness from 12 to 242 mg. In general, there is no direct correlation between the abrasiveness of coal and either its rank or its ash content in the raw state. The two strongly-caking samples of coal, Cush Creek and Wilkeson-Wingate, both gave abrasion losses of less than 50 mg. Whether or not low abrasiveness characterizes all strongly caking coal, however, scarcely can be deduced from tests of only two such coals. The Pennsylvania anthracite was

Table XIII. Results of Abrasion Tests Made on Specific-Gravity Fractions of Various Kinds of Coal

Coal	Specific Gravity	Weight, Pct	Ash,[a] Pct	Cumulative			
				Abrasion Loss, Mg.	Weight, Pct	Ash,[a] Pct	Abrasion Loss, Mg.
Wilkeson-Wingate	Under 1.30	11.5	4.9	4	11.5	4.9	4
	1.30 to 1.40	37.4	10.9	5	48.9	9.5	6
	1.40 to 1.50	12.6	19.1	30	61.5	11.5	10
	1.50 to 1.60	5.3	25.9	54	66.8	12.6	13
	Over 1.60	33.2	61.6	289	100.0	28.9	105
	Total raw coal						43
Tono	Under 1.30	41.0	5.9	17	41.0	5.9	17
	1.30 to 1.40	45.9	11.6	38	86.9	8.9	28
	1.40 to 1.50	8.7	27.8	84	95.6	10.6	39
	Over 1.50	4.4	56.6		100.0	12.7	
	Total raw coal						45
Langley No. 9	Under 1.60	90.7	9.3	45	90.7	9.3	45
	Over 1.60	9.3	88.3	1515	100.0	13.8	181
	Total raw coal						234
Pennsylvania Anthracite	Under 1.60	81.1	7.6	63	81.1	7.6	63
	Over 1.60	18.9	71.8	2847	100.0	19.0	589
	Total raw coal						686
Cush Creek	Under 1.60	92.9	5.6	6	92.9	5.6	6
	Over 1.60	7.1	62.1	351	100.0	9.6	12
	Total raw coal						12
Montour No. 10	Under 1.60	79.3	9.1	43	79.3	9.1	43
	Over 1.60	20.7	75.9	618	100.0	22.9	162
	Total raw coal						171
Castle Gate	Under 1.60	95.2	6.7	147	95.2	6.7	147
	Over 1.60	4.8	63.7	1517	100.0	9.4	213
	Total raw coal						212

[a] Moisture-free basis.

over three times as abrasive as most bituminous coal; but, as will be shown later, this abrasiveness was caused more by the character of the impurities associated with it than by the coal substance itself.

For the sake of comparison, the results of abrasion tests on crushed sandstone and coke are included with the coal in Table XII. The sandstone gave an abrasion loss of 1210 mg, and the abrasiveness of the coke was 2506 mg. Thus, both of these materials, which are highly abrasive, are shown by this test procedure to be many times more abrasive than coal.

Tests of Specific-Gravity Fractions

To determine to what extent the abrasiveness of a raw coal is caused by the character of the coal substance itself and to what extent it is caused by the impurities associated with the coal, five coal samples were separated into clean coal and impurity by float-and-sink tests at 1.60 sp gr. Abrasion tests were then made separately on the fractions. Two other samples of coal were separated at additional specific gravities to show the character of the material of intermediate density. Table XIII presents the results.

In every instance, the impurity heavier than 1.60 or 1.80 sp gr was much more abrasive than the clean coal. For example, clean Pennsylvania anthracite lighter than 1.80 sp gr gave an abrasion loss of 63 mg, while the heavy impurity gave 2847 mg. This impurity, which appeared to be slate, was the most abrasive material tested, exceeding in abrasiveness even the figure of 2506 mg for coke. The float 1.80 sp gr fraction of the Pennsylvania anthracite was less abrasive than many raw coals tested, and less abrasive than even the float fraction of the Castle Gate coal.

It will be observed that, with several of the coal samples shown in Table XIII, the cumulative abrasion loss for the specific gravity fractions tested individually differs appreciably from the abrasion loss obtained when these materials were tested combined together as raw coal. The explanation for this discrepancy is not the same for all coals. With the Langley No. 9 coal, the cumulative abrasion loss of 181 mg is thought to be less than the value of 234 mg obtained on the raw coal because of the loss of considerable clay during the float-and-sink manipulation. With the Wilkeson-Wingate coal, however, in which the cumulative abrasion loss is 105 mg in comparison with a loss of 43 mg for the total raw coal, the explanation appeared to be that, in tests of the raw coal, the wearing blades were shielded from action of the abrasive sink material by the presence of the much less abrasive clean coal.

The results presented in Table XIII for seven samples of coal demonstrate that abrasive wear can be attributed more to the impurities associated with coal than to the character of the coal substance itself. Cleaning should almost invariably reduce the abrasiveness of a coal through the removal of abrasive impurities, and clean, well-prepared coal will cover a much narrower range of abrasiveness than that exhibited by unwashed coal.

Summary and Conclusions

A laboratory procedure for estimating the abrasiveness of coal was developed, in which a set of wearing blades is revolved for a fixed number of revolutions in a charge of coal. The loss in weight sustained by the blades is taken as a measure of the abrasiveness of the coal.

As with any empirical test method, the results are influenced by test conditions, and such variables as speed and duration of rotation, weight and size of coal employed, and clearance between the wearing blades and the walls of the mortar containing the coal sample were investigated. With such variables properly controlled, the method gives results that can be reproduced within about 3 pct with abrasive coals, with variations of up to 18 pct occur in testing coals of very low abrasiveness. This degree of precision probably is sufficient for most purposes.

Fourteen coals have been tested, and seven of these were separated into clean coal and impurity by float-and-sink procedure to permit testing the coal and its associated impurities separately for abrasiveness. In every case the impurities proved far more abrasive than the coal substance itself. Clean coal ranged in abrasiveness from about 10 to 150 mg, while the heavy impurities separated from these samples ranged from about 300 to 2850 mg. Thus the amount and character of the impurities associated with a coal have a marked influence on its abrasiveness.

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H. N. Appleton, Asst. to Secy. H. A. Maloney, Asst. Treas.
John V. Beall, Eastern Secretary, Mining Branch
29 W. 39th St., New York 18
R. E. O'Brien, Western Secretary, Mining Branch
808 Newhouse Bldg., Salt Lake City

aime NEWS

Engineering Foundation Reports on Research Projects

Engineering Foundation, joint research organization of the four major engineering societies, summarized the progress on the 14 projects carried out under its sponsorship during the year ending Sept. 30, 1950. Three of these projects are in the field of mechanical engineering, five in civil engineering, four in mining and metallurgical engineering, and two are cooperative projects that cut across the whole engineering field. For the support of these projects the foundation made grants totaling $25,500, and industry and others contributed $967,045.

In the field of mechanical engineering, considerable progress has been made in increasing the accuracy of methods for measuring flowing fluids, in determining the forces needed in the rolling of metals, and in increasing the efficiency of large coal-fired boilers in power-generating plants by eliminating lazy surface, thus pointing the way to a more complete combustion of the fuel.

Much important work was done during the year in the field of mining and metallurgy. Noteworthy is the work at MIT on the self-diffusion of iron, in which radioactive iron, supplied by the Atomic Energy Commission, was used as a tracer. This work has shed new light on the structural changes that occur when steel is hardened by quenching; it has shown that these changes take place with split-second rapidity at a temperature only 4° above absolute zero. This refutes the previously advanced theory that such changes take place by atom-by-atom growth, in which case the changes would need centuries for completion at temperatures near absolute zero.

Karl T. Compton Awarded Hoover Medal

Karl T. Compton, chairman of the Corporation of Massachusetts Institute of Technology, was awarded the 1950 Hoover Medal for "distinguished public service." The award was made to the M.I.T. scientist as a "Great leader in engineering education, who has had a profound influence on the development of science and engineering, and has devoted himself wholeheartedly to the welfare of the nation, both in times of peace and in times of war . . ."

Dr. Compton is the twelfth engineer to receive the medal since it was first awarded to Herbert Hoover in 1930 to commemorate his civic and humanitarian achievements. The medal was established through a trust fund created by a gift of Conrad N. Lauer and is awarded periodically to distinguished engineers selected by a Board from the AIME, AIEE, ASME, and ASCE.

Oregon Section Host at Nonmetallic Conference

The Oregon Section is to be the host of the 1951 Northwest Industrial Minerals Conference to be held at the Congress Hotel, Portland, Ore., Apr. 27 and 28. On the 27th there will be morning and afternoon sessions at which technical papers will be presented; a luncheon, cocktail party, and banquet are also scheduled for that day. On the following day a series of plant tours are tentatively scheduled.

Change of Address

When notifying AIME headquarters of a change of address, or of company position or affiliation, *please mention the Branch of the Institute to which you belong—Mining, Metals, or Petroleum.* This will make for a more expeditious handling of the change and will facilitate the preparation of various reports.

Award Revision Approved

At the Jan. 17 meeting of the Executive and Finance Committees, approval was voted of a revision in the citation for the 1951 C. E. Johnson, Jr. Award to Samuel Naismith, which had been approved previously by the Board of Directors. The revision reads as follows: "For his contribution to the knowledge of blast furnace raw materials."

Oak Ridge Course In Nuclear Technology

Industrial organizations interested in obtaining special training in nuclear reactor technology for engineers in their employ are invited to sponsor their enrollment in the 1951-52 session of the Oak Ridge School of Reactor Technology. Application deadline is Apr. 1, and the school term begins Sept. 10, 1951. A limited number of recent college graduates are accepted in category A in the status of student employees. Category B students are selected from the applications sponsored by government agencies and private industrial firms interested in participating in the AEC national nuclear energy program. Such firms need not be connected with the AEC currently. Further information and application forms may be obtained by writing to the Director, Oak Ridge School of Reactor Technology, P. O. Box P, Oak Ridge, Tenn.

Since much of the material presented in the curriculum of the Oak Ridge School of Reactor Technology will be classified, all enrollments are contingent upon a personal security investigation. Announcements of appointments will be in April.

Annual Meeting Bituminous Coal Research

A new date has been set for the annual meeting of the Bituminous Coal Research, Inc., in Columbus, Ohio. Mar. 7 has been selected and all speakers scheduled previously have accepted the new date and the program will be the same. The meeting will be held at the Deshler-Wallick Hotel. The recent railroad situation forced postponement from the original date in February.

Drift of Things . . . . . as followed by **Edward H. Robie**

WARTIME demands for metals have focussed attention of the Government on the mining industry to a greater extent than at any time in history. Bureaus in Washington are being established and expanded daily to insure mine production of metals, nonmetallic minerals, and coal in sufficient quantities to meet any national emergency. Ways and means are constantly being sought to throw Government money into mining enterprises and expansions. It is of this latter phase that some hardheaded consideration had better be given.

Subsidies and price parities, and occasionally actual Government operation appear to be the simplest and easiest ways to boost production. The long term effect of such practices on the mining industry, however, is questionable. Subsidies can and often do grow into Frankensteins (witness egg, butter, and potato subsidies) and not only open the door but eventually pave the way to Government ownership and Government operation. There certainly must be other ways of getting maximum production on what might be termed marginal or borderline properties. There certainly must be other ways of financing that will not only insure the operator that the development will pay for itself, but will insure the nation maximum production at a cost that is as economical as possible.

Such a method conceivably could be a loan-purchase arrangement. Under this type of plan, mine operators would do the development work, operate the mine and auxiliary processes, and eventually be permitted to acquire full ownership of a property. On the other hand, the Government would be assured of maximum production at a minimum cost.

By taking a hypothetical situation, the plan works out along general lines as follows:

Cobalt is in great demand. Under a normal world economy, the metal is imported almost entirely. However, the threat of war and its effect on shipping demands that some other sources, preferably domestic, be developed. There is cobalt ore in this country. It is low grade ore, a byproduct of mining of silver or other metals, and has not been given too much attention because the cost of production makes competition with foreign metal prohibitive.

The plan for development of domestic cobalt sources would be instituted by the Government. The Government would, after proper consideration and consultation with experienced mining men, finance the construction and development of a cobalt mining and processing property, including mining, milling, processing, and anything necessary to the production of desired cobalt metal or alloys. The company chosen to operate the mine or property would then arrange for its development and construction. Actually, the financing would be in the form of a loan to the company.

From the operation of the property, the company would pay back the loan by a prearranged but definitely established method. It could be on the basis that during the operation of the property, the company would keep 40 pct or 50 pct of the profit, and the remainder would be paid back to the Government to amortize the investment.

If, on the other hand, no profit could be made on the enterprise, the Government would maintain full ownership and pay the out-of-pocket expenses of the operation. When the critical need for the material is past, the Government would maintain the property in standby condition on a contract basis, as it did with many properties after World War II. This latter condition would occur only if cobalt could not be produced by this property at a profit. As long as the project is profitable, the company would continue to operate it and amortize the investment from profits. If uneconomical and placed in standby, at any time that it is deemed necessary to produce metal, the Government could reactivate the project, and the company would get back into production.

If the entire operation is mismanaged, the Government could call the company in for a review of the entire project. If it develops that the property is mismanaged, negotiations with other companies in the field can be started. If a new company does take over the project, it should do so where the old company left off and production would continue.

The whole idea behind the plan is: (1) Insure a sufficient quantity of critical materials during any national emergency; (2) Permit these materials to be produced by mining companies; (3) Give mining companies an opportunity to acquire properties that can be operated competitively under peacetime as well as wartime economies; (4) Keep Government out of industry and head off any trend toward Government ownership and operation of industry.

Obviously, a plan of such magnitude will have many facets and details that must be worked out. However, if these details are worked out with the purpose of the plan in mind, a healthier, stronger, more aggressive mining industry will result. Obviously, not all such projects will be able to withstand peacetime competition, and will have to curtail operations eventually. This is no great handicap. Instead, it is the same as a life insurance policy. The project is there, ready to operate when needed, yet a company is not saddled with a financial liability that would threaten its very existence in normal times.

While this plan might be adaptable to mining, there is no reason why it could not be applied to any of the basic industries, including steel. New England states have been juggling the idea of a new steel plant for some time, and at present it looks as though it will be completely Government financed.

While steel enterprises have not been subsidized by the Government, the loan-purchase idea may be appealing. It would eliminate any chance of real loss to the Government. Geneva Steel Co. was sold after the war at a figure lower than the cost of construction, and Kaiser Steel Co. still has involved financial arrangements. On the loan-purchase plan, the plant belongs to the Government until it is paid for, but as long as the operation is profitable its amortization is automatic.

If a change of pace and style of writing is noted in Drift of Things this month, it was because Mr. Robie, faced with a mountain of work for the coming annual meeting, was unable to write the column. Pinch-hitting for Mr. Robie was Thomas E. Lloyd, Editor, JOURNAL OF METALS.

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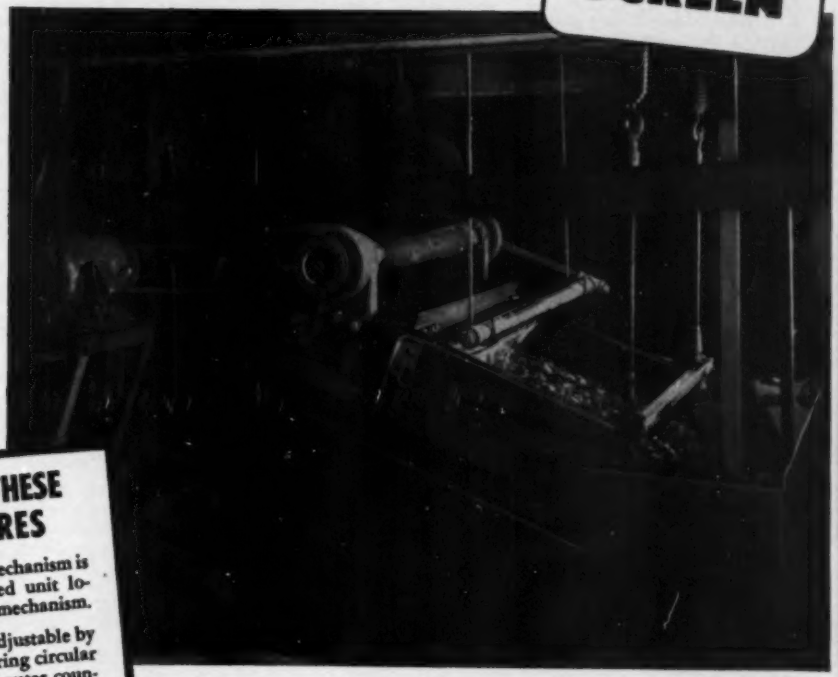

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Personals

C. A. Anderson of Prescott, Ariz., will head the geological survey section of the Defense Minerals Administration's field office at Tucson.

F. R. Archibald, formerly located at Duluth, is now connected with Ventures, Ltd., 25 King St., West, Toronto, Ontario.

E. R. Bechtel, chief chemist and supervisor of ore movement of Oliver Iron Mining Co., retired after over 42 years of service on the Mesabi range and the company general offices in Duluth.

J. Parnell Caulfield, formerly general manager of the Hudson Bay Mining & Smelting Co., Ltd. operations at Flin Flon, Canada, has resigned and is now general manager of the Utah div., Kennecott Copper Co.

L. C. Campbell, vice-president, Eastern Gas & Fuel Associates, Pittsburgh, has been appointed chairman of the Coal Div., American Mining Congress, Washington, D. C.

C. S. T. Farish has retired as general administrative superintendent of the Cerro de Pasco Corp., La Oroya, Peru and is temporarily residing at Lima.

Max W. Freeman is now petroleum field engineer with the Skelly Oil Co., Carthage, Texas.

Thomas G. Fear is now assistant chief engineer at the Clinchfield Coal Corp., Dante, Va.

JOHN FAYERWEATHER

John Fayerweather is now on active duty with the Munitions Board, Washington, D. C. He had formerly been an instructor at the Harvard Business School, Boston.

Patrick E. Green is now associated with the American Smelting & Refining Co., Santa Barbara, Chih., Mexico.

Leroy J. Harrington, former plant manager of the American Cyanamid Co., New York, has been named manager of bauxite mining.

Arthur L. Lloyd has resigned his position of mining engineer at Caledonia Collieries, Ltd., Cessrock, N.S.W., Australia, and accepted the position of assistant mine superintendent, Lake George Mines Pty., Ltd., Captains Flat, N.S.W., Australia.

R. E. McMILLEN

R. E. McMillen has been transferred to the Houston office of the Ohio Oil Co. as division engineer of the Texas div. **Fred V. Kluck,** formerly Kansas district engineer, is succeeding Mr. McMillen as division engineer of the Oklahoma-Kansas div.

Omega N. Moreland is presently employed by the Kennecott Copper Corp., Harley, N. Mex.

Roger O. Oscarson has resigned his position as secretary, treasurer, and manager of the Northwest Mining Assn. He is now director of the new field office of the U. S. Dept. of Commerce, Spokane.

Felix Ochs, consulting engineer for the Felix Kramarsky Corp., New York, has returned to the U. S. after having started the operation of a copper mine in Haiti.

Charles F. Park, Jr., dean, Stanford School of Mineral Sciences, Stanford University, Stanford, Calif., has left for a three-month trip to Japan. He will be attached to the mining and geology div. of the Natural Resources Section of General MacArthur's headquarters and will assist the staff in the modernization and rehabilitation of the Japanese mining industry.

L. Sonneveld, mining engineer for the Compania Minera Unificada del Cerro de Potosi, has been transferred from Pulacayo to Potosi, Bolivia.

John W. Sheedy has accepted the position of mill superintendent at Murgul Bakir Islet Mesi, Murgul, Turkey. He had been working with the Munitions Board, Washington, D. C.

Clarence Sleeman, engineer at the Hill-Annex mine, Calumet, Minn., of the Inter-State Iron Co. has been transferred to the firm's general mining offices at Virginia, Minn. as assistant chief engineer.

J. L. Strong, mining engineer with the Oliver Iron Mining Co., has retired. Mr. Strong joined the company following his graduation from college in 1908. He first worked at Hibbing on the Mesabi range and was then transferred to the general mining engineering offices at Duluth. He will spend a few months traveling and then reside in Duluth.

Albert O. Trepuk is president of the Douglas Export-Import Corp., 17 Battery Place, New York.

James C. Gray, formerly assistant manager of raw materials for the Tennessee Coal, Iron & Railroad Co., has been made manager of manufacturing for the company.

George C. Grow, Jr., is now manager of the underground gas storage dept., Transcontinental Gas Pipe Line Corp., Newark, N. J.

J. ROBERT THOMPSON

J. Robert Thompson has been appointed manager of the Atlanta district, industrial products sales div., B. F. Goodrich Co.

Samuel H. Williston, vice-president of Cordero Mining Co., has been appointed director of the supply div., Defense Minerals Administration.

Coming Events

Mar. 2, AIME, San Francisco Section, joint meeting with Auxiliary, Sir Francis Drake Hotel, San Francisco.

Mar. 5, AIME, Pittsburgh Section, Coal Div., Mellon Institute Auditorium, Pittsburgh.

Mar. 5, AIME, Florida Section, Lakeland, Fla.

Mar. 5-7, Hydraulic Institute, quarterly meeting, Santa Barbara Biltmore Hotel, Santa Barbara, Calif.

Mar. 5-7, Manufacturers Standardization Society of the Valve and Fittings Industry, annual meeting, Commodore Hotel, New York.

Mar. 5-9, ASTM, spring meeting and committee week, Cincinnati.

Mar. 6, Society for Applied Spectroscopy, 6 pm dinner, Tosca's; 8 pm meeting, Socony-Vacuum Training Center, New York, N. Y.

Mar. 11-14, American Institute of Chemical Engineers, regional meeting, Greenbrier, White Sulphur Springs, Va.

Mar. 13-16, National Assn. of Corrosion Engineers, conference and exhibition, Hotel Statler, New York.

Mar. 19-23, Conference on Industrial Personnel, Dept. of Industrial Engineering, Columbia University, New York, N. Y.

Mar. 21-22, Combustion of Industrial Fuels, conference; sponsorship of College of Engineering, University of Michigan, Ann Arbor.

Mar. 22, ASTM, Philadelphia District, Philadelphia.

Mar. 26-27, Natural Gas Engineering Lectures, University of Kansas campus, Lawrence, Kan.

Mar. 31-Apr. 2, AIME, Blast Furnace, Coke Oven and Raw Materials and National Open Hearth Steel Conference, William Penn Hotel, Pittsburgh.

Apr. 2, AIME, Pittsburgh Section, Oil and Gas, Mellon Institute Auditorium, Pittsburgh.

Apr. 2, AIME, Mexico Section, American Club, Mexico City.

Apr. 2-5, ASME, spring meeting, Atlanta-Biltmore, Atlanta.

Apr. 3, Society for Applied Spectroscopy, 6 pm dinner, Tosca's; 8 pm meeting, Socony-Vacuum Training Center, New York.

Apr. 3-6, Safety Convention and Exposition, Statler and Governor Clinton Hotels, New York.

Apr. 4-6, Midwest Power Conference, Sherman Hotel, Chicago.

Apr. 11, AIME, Chicago Section, Chicago Bar Assn., 29 S. LaSalle St., Chicago.

Apr. 15-18, Scientific Apparatus Makers Assn., annual meeting, Greenbrier, White Sulphur Springs, W. Va.

Apr. 16-18, American Society of Lubrication Engineers, national convention, Bellevue-Stratford Hotel, Philadelphia.

Apr. 17-20, American Management Assn., National Packaging Exposition, Auditorium, Atlantic City, N. J.

Apr. 20-21, Engineers' Day, Colorado School of Mines, Golden.

Apr. 23-26, American Foundrymen's Society, annual convention, Buffalo, N. Y.

Apr. 25-26, Metal Powder Assn., annual meeting, Hotel Cleveland, Cleveland.

Apr. 30-May 1, Assn. of Iron and Steel Engineers, spring conference, Hotel Statler, Detroit.

Apr. 30-May 4, Materials Handling Exposition, International Amphitheatre, Chicago.

Apr. 30-May 11, British Industries Fair, annual industrial show, Olympia and Earls Court, London; Castle Bromwich, Birmingham.

May 4-9, Anthracite Institute, annual meeting, Lehigh University, Bethlehem, Pa.

May 7, AIME, Mexico Section, American Club, Mexico City.

May 7, AIME, Florida Section, Lakeland, Fla.

May 7-11, Greater New York Industrial Show, 71st Regiment Armory, New York, N. Y.

May 9-11, Engineering Institute of Canada, annual meeting, Mount Royal Hotel, Montreal.

May 15-16, American Institute of Chemical Engineers, regional meeting, Hotel Muehlebach, Kansas City, Mo.

May 14-17, American Mining Congress Coal Mining Convention & Exposition, Auditorium, Cleveland.

May 23-24, American Society for Quality Control, annual convention, Hotel Cleveland, Cleveland.

June 11-15, Conference on Industrial Research, Columbia University, New York.

June 11-15, ASME, semi-annual meeting, Royal York, Toronto, Canada.

June 18-22, ASTM, annual meeting, Chalfonte-Haddon Hall, Atlantic City, N. J.

Sept. 16-19, American Institute of Chemical Engineers, regional meeting, Sheraton Hotel, Rochester, N. Y.

Sept. 25-28, ASME, fall meeting, Minneapolis.

Oct. 1-4, Assn. of Iron and Steel Engineers, annual convention, Hotel Sherman, Chicago.

Oct. 3-5, AIME, Petroleum Branch, fall meeting, Oklahoma City.

Oct. 8-12, National Hardware Show, Grand Central Palace, New York, N. Y.

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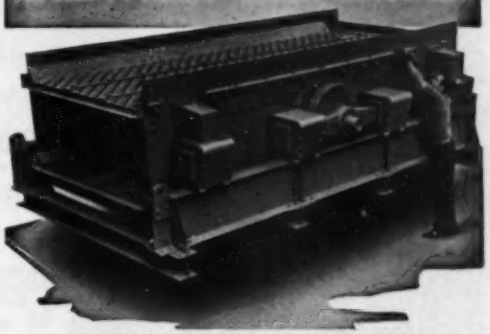

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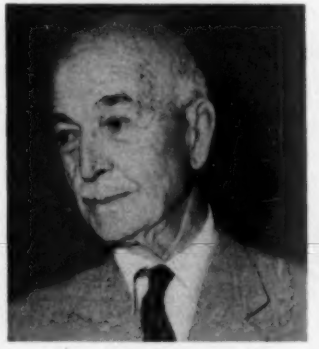
Appreciation of James Ralph Finlay

By Heath Steele

James Ralph Finlay was born in Blenheim, Ont., Canada, on Sept. 30, 1869. He was the son of Ralph Spence Finlay and Anna Rankin, both of Scotch-Irish descent, having moved to Canada from the Londonderry district of Ireland. When he was nine his parents died and he went to live with an uncle at Colorado Springs. He attended the Colorado College at Colorado Springs and in 1891 received an A.B. from Harvard. On Aug. 10, 1904 he married Edith Davis Adams in Spokane, Wash. Two sons, Phillips and Madison, reside in Southern California.

Finlay was on the Harvard crew during 1888-89 and the football eleven in 1890; he was on the team that beat Yale for the first time in 15 years. He was on the track team in 1891 and broke two Harvard records. The first was in April 1891 when he threw the hammer a distance of 94 ft 8 in. The second was in May of 1891 when he threw the hammer a distance of 108 ft 5 in. At the age of 70 he shot a 72 in golf at Westhampton Beach, Long Island, winning the club championship.

While attending Harvard, Finlay worked in the mines of Colorado during vacations and also examined some gold properties in Nova Scotia. He also worked with geological survey parties in Colorado during vacations. In 1891 he was employed by the Michigan Land & Iron Co. to run a geological survey in the Upper Peninsula in a search for iron ore. He did exploration work on the



J. R. FINLAY

Mesabi Iron Range and discovered an iron float which afterwards proved to be exactly over the great Sauntry-Alpens Mine of the Oliver Iron Co. In 1894 he worked as mining engineer for the Minnesota Hard Ore Mine at Tower, Minn. Later he became superintendent of the Norman mine. In 1896 he returned to Colorado, and worked for a while as a miner at the Isabel mine. Later he was engaged in a rather extensive prospecting trip in Colorado. In 1897 he went to Ecuador to work with the Twombly interests in the Zaruma mine, later known as the Portovello mine, which has just reached the end of its production period. Later Finlay was engaged in work for the Cleveland-Cliffs Iron Co. Afterwards he became superintendent of the Standard Mines at Mace, Idaho, and in 1902 he was assistant manager of the Bunker Hill & Sullivan mines under Burch. His first real job was as manager of the Portland Gold Mining Co. at Cripple Creek, Colo. It was here that he became convinced of the fallacy of working for a low cost per ton, and, by eliminating the mining of waste grade, increased the profits of the Portland mine. In 1903 he was in southeast Missouri as assistant superintendent of the Federal Lead Co., a Guggenheim property, which, while comparing favorably with other lead properties of the district, was steadily losing money. Again Finlay insisted upon clean mining and placed the property on a profitable basis. From 1904 to 1907 he was engaged in consulting work, with offices first in Colorado Springs and later in New York City. His examinations carried him to every state in the Union, Canada, the mining regions of South Africa, Australia, and other parts of the world. In 1909 he became general manager of the Goldfields Consolidated Mines at Goldfield, Nev.

In 1911, Mr. Finlay went to Michigan to appraise the iron and copper mines for the State Tax Commission. After his work there he again established a consulting office in New York. He was later engaged in appraising the mining properties of New Mexico for the State, and did considerable work for Phelps Dodge in Arizona. About 1917 he made his headquarters in the West and finally became a resident of Redlands, Calif.

Finlay was the author of *The Cost of Mining* in 1909 and 1919. For several years he lectured at Harvard and was also the author of many technical articles on mining and geology.

To express my own feelings regarding "J.R.", I can do no better than to quote parts of a letter I received from Professor L. C. Graton who first met Finlay in 1903 at the Portland mines in Cripple Creek: "I have never known anyone who impressed me more as possessing absolute integrity. It was manifested constantly by what he said and the way he reacted. . . . Finlay was a man of

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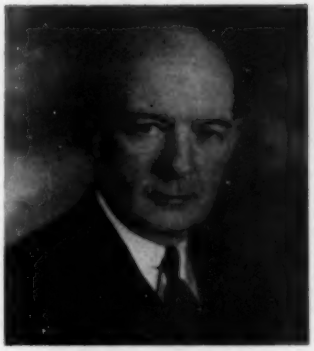
outstanding stature, intellect and character, who sensed what is sound and wholesome and sought to promote it, who recognized what is wrong and dangerous and spoke out clearly against it; and who, possessed of a simple nature, held and revealed a friendly regard for those whom he felt worthy. . . . The high esteem in which he was widely held reflects as creditably on our profession as on him."

Royer E. Galbreath (Member 1921) died on July 8, 1950. Born in Johnstown, Pa. in 1887, he attended school there. In 1909 he joined the Cambria Steel Co., Johnstown, as construction engineer. For three years he was connected with the Vintoy Colliery Co., Vintondale, Pa. He joined the Wisconsin Steel Co., coal and coke works, a subsidiary of International Harvester Co., Benham, Ky. in 1918 and from 1921 to 1929 he was in charge of operations. He was made superintendent in 1940 and held that position at the time of his death.

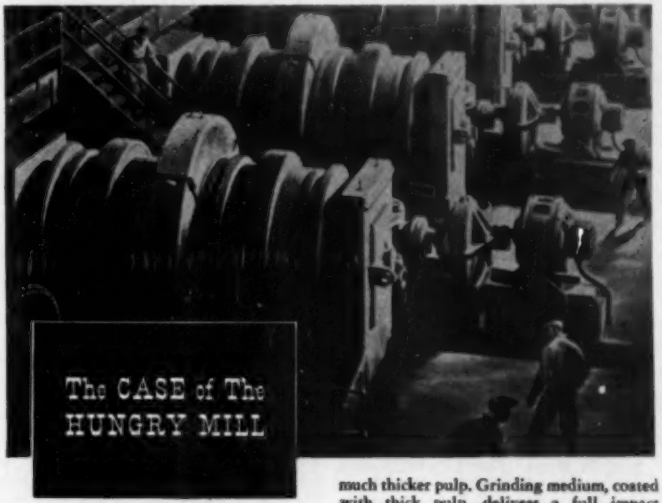
Fred S. Gallus (Member 1920) died of a heart attack on Dec. 16, 1950. Mr. Gallus was born in Crissona, Pa. on Mar. 9, 1886. In 1908 he joined the Dodson Coal Co., Morea, Pa. as assistant engineer and then became engineer. He later was employed by the Alliance Coal Mining Co., Kaska, Pa. and in 1915 joined the Pine Hill Coal Co., Minersville, Pa. as assistant superintendent in charge of mining. He remained with them for one year and then became associated with the Lehigh Coal & Navigation Co., Lansford, Pa. He was district engineer, district stripping superintendent, and district superintendent, respectively.

A. R. Gordon (Member 1911) died on Jan. 7, 1951. Born in Chicago in 1882, he attended Armour Institute of Technology, Chicago, for two years studying civil engineering. He also attended Michigan College of Mines studying mining. He went to Honduras in 1907 as mine superintendent for the N. Y. & Honduras Rosario Mining Co. In 1915 he became general manager and then vice-president. He returned to the U. S. in 1936 and in 1937 opened his office as consulting mining engineer in Washington, D. C. Mr. Gordon was a Life Member of AIME.

Arthur Crandell Green (Member 1921), former director and vice-president of sales, Goodman Mfg. Co., died on Oct. 31, 1950. He was born in Clarkston, Mich., in 1882 and was educated at the University of Michigan. In 1905 he graduated with the degree of B.S. In 1904 he was employed by the Bucyrus Co., Milwaukee as a draftsman, and for a short time held the same position with Allis-Chalmers. One year was spent doing drafting and construction work for the Arizona Smelting Co. In 1906 he joined the Balback Smelting & Refining Co. as chief engineer. He became associated with the American Smelting & Refining Co. again engaged in construction and drafting work. Some time was spent in Borneo as resident engineer for the Malayrian Rubber Co. He returned to the U. S. in 1910 as manager for Great Falls Iron Works. In 1911 he joined the Goodman Mfg. Co., Chicago, as sales engineer and district manager. In 1930 he was appointed central district manager and in 1936 was given charge of western sales and elected director of the company. He became general sales manager in 1937 and two years later was made vice-president and sales manager.



ARTHUR C. GREEN



Two grinding mills running side by side . . . one accepting only moderate amounts of ore but always overfull, never turning out the tonnage expected of it. The second mill, grinding ravenously, always delivering well over its capacity rating. The size was the same, construction made the difference.

Conventional trunnion overflow mills run nearly half full at normal speeds, with a balanced load condition which uses only part of the power made available by the driving motor. MARCY Low Pulp Line mills have a higher rate of discharge, use a much thicker pulp. Grinding medium, coated with thick pulp, delivers a full impact drop with faster reduction. The load in a MARCY is never balanced, always uses the full torque of the driving motor.

In the case quoted here, the second mill, a MARCY, paid off in 29.6% faster grinding and proportionately lower costs per square foot of floor space. This is a true story taken from records of operating milling plants. Ask us to give you complete facts on MARCY grinding under your conditions. Our engineers can show you how MARCY mills will save you money in many ways. Write for our free engineering services.

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THE Mine & Smelter SUPPLY COMPANY

Obituaries

(*Continued*)

Abraham S. Hersov (Member 1941) died suddenly in February 1949. Mr. Hersov was born at Cape Town, South Africa in 1895 and in 1914 received the degree of B.Sc. from the University of Johannesburg. During World War I he was stationed with the military forces in East Africa. After the war he was surveyor for the Knight Central G. M. Co. Ltd. In 1922 he was partner and surveyor with Nillsen & Hersov. In 1934 he became chairman and managing director of Anglo Transvaal Consolidated Investment Co., Ltd., a group of gold and base mineral mining companies. At the time of his death he was still located in Johannesburg, Transvaal, S. Africa.

Charles W. Howbert (Member 1938) died on Dec. 4, 1950 after a short illness. Mr. Howbert was born at Clarinda, Iowa and spent most of his life at mining positions in the western part of the country. In 1895 he was superintendent of Anchoria-Leland Mining & Milling Co., Colo. and in 1897 he was made general manager, holding this position for ten years. Four years were spent in Sonora, Chihuahua, and Michoacan, Mexico, doing mining and prospecting work. In 1911 he became general manager for the Ethel Louise Gold Mining Co., Cripple Creek district, Colo. He was general manager for several mining concerns all located in the Cripple Creek district for approximately 20 years and in 1935 joined the Conundrum Gold Mining Co. as general manager, a position he held at the time of his death.

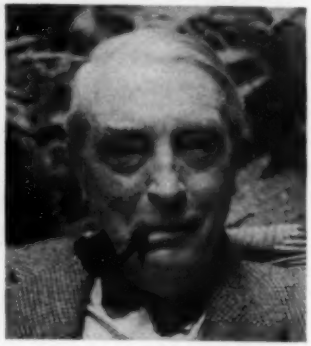
Karl F. Klein (Member 1931) died on Jan. 21, 1951. Born in New York City, he graduated from Columbia University, receiving the degree of E.M. in 1916. Following graduation he went to South America and was employed by several mining companies in Peru and Chile. In 1921 he joined the British Equatorial Oil Co., Venezuela as engineer. He returned to the United States and joined the Ingersoll-Rand Co., New York. In 1925 he returned to Chile for the International Machine Co. He went to Colorado in 1932 and became general superintendent for the Cia. Minera Carlota, Las Melosas, Chile in 1935. He joined the Andes Copper Mining Co., Potrerillos in 1938 and remained with them for several years. At the time of his death he was employed by the Anaconda Mining Co. in South America.

Harry V. Long (Member 1940) was killed in a flood that swept the mine at which he was employed on Oct. 16, 1950. Mr. Long was born at Manhattan, Kan. in 1919 and graduated from McGill University, Montreal with the degree of B.Eng. in 1940. While attending college he had several months' experience working with mining companies located in Canada. Following graduation he remained in Canada for a short time and in 1942 went to Nicaragua as mill superintendent for the Empresa Minera de Nicaragua. During World War II he was in the Royal Canadian Air Force. After his discharge he returned to Nicaragua and joined the Compania Minera La India. He was field engineer, mine superintendent, and then general superintendent for the same concern.

William M. Myers (Member 1938) died on Jan. 25, 1951. Mr. Myers was born in Clinton, N. Y. in 1892. He attended Syracuse, Columbia, and Michigan Universities and in 1933 received the degree of Ph.D. Following graduation he was employed as a chemist by various concerns. He did some metallurgical work in the analysis of smelter and mine and mill operations. In 1919 he joined the New Cornelia Copper Co., Ajo. He was employed by various companies and in 1921 became mineral technologist for the U. S. Bureau of Mines, nonmetallic station. He later joined the faculty of Pennsylvania State College, teaching mineral economics and technology.

Kenneth M. Quickel (Member 1942) died on Dec. 4, 1950. Born at Harrisburg, Pa., Mr. Quickel graduated from Pennsylvania State College in 1911 with the degree of B.S. in E.M. For a short time he was employed as an engineer for the Montlake Coal Co., Chattanooga, and then joined the Pond Creek Coal Co., Stone, Ky., as chainman, transitman, and division engineer. In 1914 he accepted the position of chief engineer with the Carbon Coal Co., Carbon, Ky. Two years later he joined Bethlehem Steel Corp., Industrial Collieries Corp., a subsidiary. While with this concern he was assistant engineer, chief engineer, special engineer, assistant to general manager, assistant general manager, general manager, and at the time of his death was manager.

Edgar Rickard (Member 1898) died in San Francisco on Jan. 21, 1951. Mr. Rickard was an associate of former President Herbert Hoover aiding him on various relief projects. Mr. Rickard was born in Pontgibaud, France. At the age of seven he came to Nevada with his father. In 1895 he graduated from the University of California with a degree in mining engineering. He practiced his profession in the United States, Alaska, Tasmania, and Mexico until 1905. Then he became business manager of the *Mining and Scientific Press* at San Francisco. In 1909 he helped start the *Mining Magazine* in London. While in London he met Mr. Hoover and during World War I they both became members of the Commission for Relief in Belgium. When Mr. Hoover returned to the United States Mr. Rickard came with him and took charge of Mr. Hoover's business interests. Mr. Rickard was assistant food administrator and in



EDGAR RICKARD

1918 was acting administrator during Mr. Hoover's absence in Europe. In 1920 Mr. Rickard became a director or officer of various manufacturing concerns, including Hazeltine Corp., Pitney-Bowe, Inc. and Pejepscott Paper Co. He was a Life Member of AIME.

George B. Rodgers (Member 1903) died of a heart condition on Sept. 10, 1950. Born in Knoxville in 1876, he attended St. Louis Manual Training School and University of Tennessee. For a few years he was employed by the Telephone Co. and then a bank in Knoxville. He joined a gold mining concern in Oregon and was superintendent and engineer of construction and assayer. In 1902 he joined the Union Electric Light & Power Co., St. Louis. He was superintendent and engineer of subways, construction dept. He also did some work in lead mines, concentrating plants, and electric power plants and accessory equipment. In 1920 he came to New York but in 1923 went to Tulsa. In 1938 he returned to St. Louis and was living there at the time of his death.

Arthur B. Shutts (Member 1911) died on Nov. 20, 1950. Mr. Shutts was born in 1880 and graduated from Northwestern and Leland Stanford Universities. He practiced mining at Grass Valley and Bodie, Calif., and Nome, Alaska. In 1911 he was manager for the Santa Mario (Mexico) Mines, Ltd., Minillas, Zacatecas, Mexico. He returned to the United States and in 1928 was assistant superintendent of the Bridgeport mine for the H. C. Frick Coke Co., S. Brownsville, Pa. A year later he joined the Pennsylvania Coal Co. in the same position. In 1936 he became associated with the Kehoe Berge Coal Co., Pittston, Pa. as general superintendent. He later moved to Bloomsburg, Pa., where he was treasurer of Western Lehigh Coal Co., Inc.

Appreciation of F. A. Thomson

By A. E. Adami

Francis A. Thomson (Member 1902), retired president of Montana School of Mines, passed away at Spokane on Jan. 11 at the age of 71. His death came but a few months after his retirement from active service when he moved his residence to Spokane. Born in London, England in 1879, he came to America at an early age and lived in British Columbia. At the age of 16 he became interested in minerals and mining, first as a prospector, miner and then an assayer. After a few years he registered at the Colorado School of Mines where he received the degree of Engineer of Mines in 1904. He did post graduate work at the Royal School of Mines in London and at the Ecole des Mines in Paris for a short period, after which he returned to the United States and worked for several mining companies in Colorado and Nevada. In the fall of 1907, he became head of the dept. of mining engineering at the State College of Washington at Pullman, Wash. He remained there until 1917 when he was named dean of the School of Mines at the University of Idaho and director of the Idaho Bureau of Mines and Geology. In 1914 he received a Master of Science degree at Golden; in 1923 his alma mater conferred on him the degree of Doctor of Science; and in 1950 the gold medal for distinguished service to his alma mater. In 1928, he was named president of Montana School of Mines and director of the Montana Bureau of Mines and Geology, which position he held until he retired from active duty on Sept. 15, 1950. Dr. Thomson was also a consulting engineer for the U. S. Bureau of Mines 1919-1928; was an active member of the AIME and at one time was a director of the society.

His passing is cause for much sorrow to the alumni and students at Montana School of Mines as well as to his colleagues and many friends in all walks of life.

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NECROLOGY

Date Elected	Name	Date of Death
1895	James R. Finlay	Jan. 1, 1951
1921	Royer E. Galbreath	July 8, 1950
1920	Fred S. Gallus	Dec. 16, 1950
1911	A. R. Gordon	Jan. 7, 1951
1931	Karl F. Klein	Jan. 21, 1951
1925	J. P. Leemans	Jan. 13, 1951
1944	Stanley M. McDonald	Dec. 11, 1950
1938	W. M. Myers	Jan. 23, 1951
1915	Clarence B. Osborne	Mar. 15, 1950
1935	William George Polisson	Dec. 13, 1950
1942	Kenneth M. Quicksi	Dec. 4, 1950
1937	Louis Roark	Unknown
1896	Edgar Rickard	Jan. 21, 1951
1936	Mark A. Smith	Dec. 29, 1950
1909	Donald Steel	Jan. 10, 1951
1947	Henry Thies	Jan. 11, 1951
1902	Francis A. Thomson	Jan. 11, 1951
1924	Alvin D. Whitman	Unknown
1911	Charles C. Whittier	Dec. 18, 1950
1907	Clarence Woods	Nov. 6, 1950

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Jan. 31, 1951, was 17,223; in addition 4613 Student Associates were enrolled.

ADMISSIONS COMMITTEE

Albert J. Phillips, Chairman; George B. Cortese, Lloyd C. Gibson, Ivan Given, Plato Malozemoff, R. D. Mollison, John T. Sherman, Harold K. Work. Alternates: Harry P. Croft, Fred W. Hanson, T. D. Jones, Frank A. Ayer, H. W. Hilzrot, and H. K. Masters.

Institute members are urged to review this list as soon as the issue is received and immediately write the Secretary's Office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate.

Alabama
Birmingham—Breland, Rufus A. (J) (C/S—S-J)
Birmingham—Bynon, George (M)
Birmingham—Holmes, Clyde T. (J) (C/S—S-J)
Birmingham—Lanier, Monro B. (J) (C/S—S-J)
Russellville—Pickard, Hardy O. (M) (R. C/S—J-M)

Arizona
Bisbee—Goar, William (A)
Ajo—Seaberg, Eaner T. (M)

California
Grass Valley—Keenan, John C. (J) (C/S—S-J)
Taft—Fleming, Frederick A. (J) (C/S—S-J)
Tecopa—Melbye, Charles E. (J) (C/S—S-J)
Whittier—Hamner, Jordan A. (M) (R. C/S—J-M)

Colorado
Cripple Creek—Bebee, Alfred Henry, Jr. (M) (R. C/S—S-M)
Cripple Creek—Carlton, Charles Henry (M) (R. C/S—S-M)
Cripple Creek—Moulton, Wesley Gunn (M) (R. C/S—S-M)
Victor—Kell, Howard Russell (M)

Delaware
Wilmington—Keane, Le Roy (A)

Idaho
Kellogg—Roberts, Irving C. (J) (C/S—S-J)
Silverton—Campbell, George B. (J) (C/S—S-J)
Wallace—Bellows, Clyde R. (M)
Wallace—Harper, Herbert E. (M) (C/S—J-M)
Wallace—MacKenzie, Wayne O. (M) (R. C/S—S-M)
Wallace—Miller, Robert A. (J)

Illinois
Chicago—Miller, James R. (J) (C/S—S-J)
Quincy—Rieman, Cedric Harlow (M)

Iowa
Des Moines—Chesebrough, Samuel A. (J) (C/S—S-J)
Fort Dodge—Fuller, Robert M. (J)

Kentucky
Closplint—Mercer, Henry C. (M)
Greenville—Whitmer, Robert D. (J) (C/S—S-J)
Jenkins—Dittrick, Alden C. (M)
McRoberts—Dann, J. C. (M)

Maryland
Hyattsville—Ellertsen, Nils A. (M) R.

Michigan
Alabaster—Connett, Robert P. (M) (C/S—J-M)
Detroit—Piper, Thomas B. (J)
Ironwood—Zuraw, Joseph (M)
Lincoln—Chapman, Rodger H. (J) (C/S—S-J)

Minnesota
Crosby—Zonteili, Emil A. (M)
Duluth—Joppa, Everett L. (M)
Duluth—Thorsen, Lloyd A. (A)
Mt. Iron—Marincel, George M. (J), C/S (C/S-S-J)

Missouri
Joplin—Rakowsky, Russell (M)
Kansas City—Booth, Stephen J., Jr. (J)
St. Louis—Dodd, Richard (M)
Webster Groves—Lake, Wilbert W., Jr. (J) (C/S—S-J)

Montana
Butte—Greene, Thomas A. (M) (C/S—J-M
Butte—Lawrence, Edmond F. (M) (C/S—J-M)
Philipsburg—Meschter, Daniel Y. (J) (C/S—S-J)

Nevada
Reno—Muller, Leopold Frederic (J) (C/S—S-J)

New York
New York City—Merriss, Philip R. (M) (C/S—J-M)

North Carolina
Monroe—Houston, Robert S. (J) (C/S—S-J)

Ohio
Shaker Heights—Bahen, John W. (A)

Oklahoma
Duncan—Callahn, James A. (J) (C/S—S-J)
Miami—Pheeps, Fred H. (M)
Miami—Wagner, Francis E. (M)

Oregon
Portland—Hogenson, Glenmore M. (J) (C/S—S-J)

Pennsylvania
Allentown—Petska, Albert M. (J)
Catasauqua—Burnell, Henry J. (J) (C/S—S-J)
Finleyville—Norton, Charles L., Jr. (J) (C/S—S-J)
Lebanon—Bail, Samuel S., Jr. (M) (R. C/S—S-M)
Mt. Carmel—Maginnis, Joseph B. (M)
State College—Saltman, Robert D. (J) (C/S—S-J)

South Dakota
Lead—Madsen, Roger A. (J)
Rapid City—Gardner, Frank (M)

Tennessee
Columbia—Shaffer, Robert B. (J) (C/S—S-J)
Isabella—Menze, Richard J. (J) (C/S—S-J)

Utah
Salt Lake City—Anderson, Donald O. (A)
Salt Lake City—Goodner, Ernest F. (M)
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Vermont
Middlebury—Brown, Phillip K. (J) (C/S—S-J)
Morrisville—Wiser, John P. (M)

Virginia
Keokee—Strang, E. Nelson, Jr. (J) (C/S—S-J)

Washington
Pullman—Dayton, Stanley H. (J) (C/S—S-J)

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Potosi—Tapia, Carlos R. (M)

Brazil
Sao Paulo—Melcher, Geraldo (J) (C/S—S-J)

Canada
Kimberley—Hodgson, Stanley (J) (C/S—S-J)
Quebec—Jerome, James E. (M) (C/S—J-M)
Yellowknife—Pinsky, George, Jr. (J) (C/S—S-J)

Chile
Potrerillos—Swensen, William T. (M) (C/S—S-M)

Cuba
Orienti Province—Zaalberg, Philippe H. A. (M)

Ireland
Sligo—Jennings, Austin G. (M)

Mexico
Mexico City, D.F.—Fries, Carl, Jr. (M) (C/S—A-M)
Parral, Chihuahua—Clark, William G. (M) (C/S—J-M)

Newfoundland
St. John's—John, John S. (M)

Peru
Diez—Canseco, Ernesto Y. (M) (R.)

South Africa
Johannesburg—Preller, Gustav S. (M) (C/S—J-M)

South West Africa
Tsumeb—McCue, J. (M)

Turkey
Zonguldak—Heike, Adolf (M) (R.)

Venezuela
Caracas—Applegate, Albert V. (J) (C/S—S-J)

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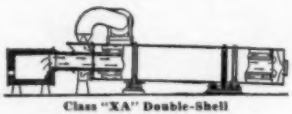

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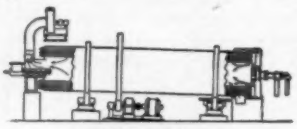

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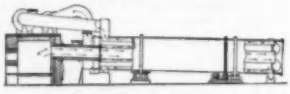

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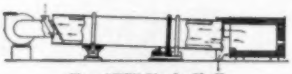

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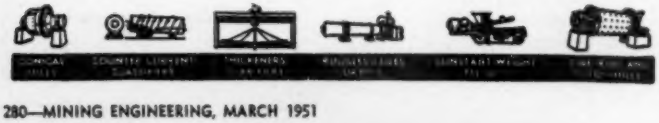

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